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TRANSACTIONS

OF THE

AMERICAN INSTITUTE OF MINING ENGINEERS.

VOL. XLIII.

CONTAINING THE PAPERS AND DISCUSSIONS OF THE NEW YORK MEETING, FEBRUARY, 1912, AND THE PAPERS OF THE NEW YORK AND SPOKANE LOCAL SECTIONS.

NEW YORK, N. Y.:
PUBLISHED BY THE INSTITUTE,
AT THE OFFICE OF THE SECRETARY.
1913.

PREFACE.

THE proceedings, papers, and discussions published by the Institute during the year 1912 were so many and so extensive that it was impracticable to include all in one regular volume without increasing the size beyond reasonable limits. It was therefore decided to issue two volumes for the publications of 1912,—Volume XLIII., to contain the papers and discussions of the New York meeting in February, 1912, and of the meetings of the New York Local Section and the Spokane Local Section; and Volume XLIV., to contain certain papers and discussions of the joint meeting of the Institute with the Eighth International Congress of Applied Chemistry, New York, September, and of the Cleveland meeting, October, 1912.

The present Volume, XLIII., accordingly contains all the proceedings, papers, and discussions of the New York meeting and the meetings of the Local Sections at New York and Spokane, 1912, with the following exceptions:

1. Brief obituary notices of members and associates reported as deceased during the year 1912; library accessions and requirements; notices of meetings of the Institute and of other societies; lists of proposed members and associates; changes of address of members; and other announcements of general but temporary interest, furnished to members in *Bulletin* Nos. 61 to 72.

2. Abstracts of lectures and descriptions of excursions and entertainments in connection with the meetings of the Local Sections, which have appeared in the *Bulletin* from time to time during the year.

3. Notices of the Joint Meeting of the Institute with the Eighth International Congress of Applied Chemistry, New York, September, 1912, *Bulletin* No. 68, August, pp. viii., ix.; and *Bulletin* No. 70, October, p. xiii.

4. The Year Book containing the List of Members and Associates revised to Feb. 1, 1912; issued directly after the Annual Business Meeting in February, 1912, in order to have the period covered correspond to the official year of the Institute.

On the other hand, this volume contains the following papers presented at the San Francisco meeting, but omitted from Volume XLII. on account of lack of space:

The Mining Industry of Japan, by Keijiro Nishio, Tokyo, Japan.

Mineral Production and Resources of China, by Thomas T. Read, San Francisco, Cal.

The Black Mountain Coal-District, Kentucky, by J. B. Dilworth, Philadelphia, Pa.

The Geology of the Tonopah Mining-District, by Augustus Locke, Goldfield, Nev.

A Modification of the "Gay Lussac" Method for Silver-Bullion Containing Tin, by Luis Emlynn Salas, New York, N. Y.

Notes on the Laramie Tunnel, by D. W. Brunton, Denver, Colo.

The Laws of Igneous Emanation Pressure, by Blamey Stevens, New York, N. Y.

Physical Data of Igneous Emanation, by Blamey Stevens, New York, N. Y.

Gold-Mines in Southern Colombia, by F. Pereira Gamba, Tuquerres, Colombia.

JOSEPH STRUTHERS,
Secretary and Editor.

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For the year ending February, 1913.

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(Term expires February, 1913.)

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E. L. YOUNG.....NEW YORK, N. Y.

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SECRETARY EMERITUS OF THE COUNCIL.

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JAMES GAYLEY, CHARLES KIRCHHOFF, FRANK LYMAN.

(Term expires February, 1913.)

JAMES DOUGLAS, JAMES F. KEMP, ALBERT R. LEDOUX.

(Term expires February, 1914.)

EDMUND B. KIRBY, CHARLES F. RAND, GEORGE C. STONE.

(Term expires February, 1915.)

Consulting Attorneys, Blair & Rudd, New York, N. Y.

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* SECRETARY'S NOTE.—The Council is the professional body, having charge of the election of members, the holding of meetings (except business meetings), and the publication of papers, proceedings, etc. The Board of Directors is the body legally responsible for the business management of the Corporation.

OFFICERS ELECTED AT ANNUAL MEETING, FEB. 18, 1913.

The list of officers on the preceding page is for the year 1912, the period covered by the contents of this volume of the *Transactions*. But the result of the election at the Annual Business Meeting, February, 1913, although strictly belonging to the next volume, is here published for the convenience of members.

The following officers were elected by vote of the members and associates in person or by proxy at the Annual Meeting, Feb. 18, 1913:

PRESIDENT.

CHARLES F. RAND, ... New York, N. Y.
(Term expires February, 1914.)

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(Term expires February, 1914.)

JAMES F. KEMP, ... New York, N. Y.
(Term expires February, 1915.)

FIRST VICE-PRESIDENT.

BENJAMIN B. THAYER, ... New York, N. Y.
(Term expires February, 1914.)

TREASURER.

GEORGE C. STONE, ... New York, N. Y.
(Term expires February, 1914.)

SECRETARY.

BRADLEY STOUGHTON, ... New York, N. Y.
(Term expires February, 1914.)

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WALDEMAR LINDGREN, District 1, . . Boston, Mass.
(Term expires February, 1914.)

BENJAMIN B. THAYER, . District 0, . . New York, N. Y.
SIDNEY J. JENNINGS, . District 0, . . New York, N. Y.
(Term expires February, 1915.)

THOMAS H. LEGGETT, . District 0, . . New York, N. Y.
FRED. W. DENTON, . District 4, . . Painesdale, Mich.
(Term expires February, 1916.)

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JAMES GAYLEY, . . District 0, . . New York, N. Y.
ALBERT R. LEDOUX, . District 0, . . New York, N. Y.
CHARLES W. MERRILL, District 6, . . San Francisco, Cal.
C. SNELLING ROBINSON, District 3, . . Youngstown, Ohio.
(Term expires February, 1914.)

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JOSEPH A. HOLMES, . District 9, . . Washington, D.C.
ROBERT W. HUNT, . District 3, . . . Chicago, Ill.
GEORGE C. STONE, . District 0, . . . New York, N. Y.
EDWARD L. YOUNG, . District 0, . . . New York, N. Y.
(Term expires February, 1915.)

JOHN W. FINCH, . District 7, . . Denver, Colo.
JOHN H. JANEWAY, JR., District 0, . . New York, N. Y.
EDWARD P. MATHEWSON, District 5, . . Anaconda, Mont.
JOSEPH W. RICHARDS, . District 2, . . So. Bethlehem, Pa.
L. D. RICKETTS, . . District 1, . . Cananea, Son., Mex.
(Term expires February, 1916.)

PAST OFFICERS.

Presidents.

*David Thomas 1871
R. W. Raymond 1872–1874
*A. L. Holley 1875
*Abram S. Hewitt 1876
*T. Sterry Hunt 1877
*Eckley B. Coxe 1878–1879
*William P. Shinn 1880
*William Metcalf 1881
*Richard P. Rothwell 1882
Robert W. Hunt 1883
James C. Bayles 1884–1885
Robert H. Richards 1886
*Thomas Egleston 1887
William B. Potter 1888
Richard Pearce 1889
*Abram S. Hewitt 1890
John Birkinbine 1891–1892
H. M. Howe 1893
*John Fritz 1894
*J. D. Weeks 1895
E. G. Spilsbury 1896
*Thomas M. Drown 1897
Charles Kirchhoff 1898
James Douglas 1899–1900
E. E. Olcott 1901–1902
Albert R. Ledoux 1903–1904
James Gayley 1905
Robert W. Hunt 1906
John Hays Hammond 1907–1908
D. W. Brunton 1909–1910
Charles Kirchhoff 1911
James F. Kemp (Council and Corporation) 1912
James Gayley (Corporation) 1905–1911

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*Martin Coryell 1871–1872
*Thomas M. Drown 1873–1884
R. W. Raymond 1884–1911
Joseph Struthers 1911–1912

Treasurers.

J. Pryor Williamson 1871–1872
*Theodore D. Rand 1872–1903
Frank Lyman 1903–1912

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* Deceased.

HONORARY MEMBERS.

Year of Election.		
1876.	Prof. Richard Åkerman	Stockholm, Sweden.
1909.	Prof. Richard Beck	Freiberg, Germany.
1905.	Andrew Carnegie	New York, N. Y.
1906.	Dr. James Douglas	New York, N. Y.
1888.	Prof. Haton de la Goupillière	Paris, France.
1906.	Sir Robert A. Hadfield	London, England.
1888.	Prof. Hans Hoefer	Leoben, Austria.
1905.	Prof. Henri Louis Le Chatelier	Paris, France.
1909.	Alexandre Pourcel	Paris, France.
1911.	Dr. Rossiter W. Raymond	New York, N. Y.
1911.	Prof. Robert H. Richards	Boston, Mass.
1909.	Dr. Ing. H. C. Emil Schroedter	Düsseldorf, Germany.
1906.	John E. Stead	Middlesbrough, England.
1909.	James M. Swank (Associate)	Philadelphia, Pa.
1902.	Prof. Dimitry Constantin Tschernoff	St. Petersburg, Russia.
1910.	Prof. Tsunashiro Wada	Tokyo, Japan.
1907.	Charles D. Walcott	Washington, D. C.

Honorary Members (*Deceased*).

		Year of Decease.
1872.	Bell, Sir Lowthian	1904
1892.	Castillo, A. Del	1895
1902.	Contreras, Manuel Maria	1902
1888.	Daubrée, A	1896
1884.	Drown, Thomas M	1904
1890.	Gaetzschmann, Moritz	1895
1873.	Gruner, L	1883
1891.	Kerl, Bruno	1905
1895.	Le Conte, Joseph	1901
1891.	Lesley, J. P.	1896
1899.	Osmond, Floris	1912
1890.	Patera, Adolph	1890
1886.	Percy, John	1889
1888.	Posepny, Franz	1895
1884.	Richter, Theodor	1898
1899.	Roberts-Austen, W. C	1902
1890.	Serlo, Albert	1898
1880.	Siemens, C. William	1883
1872.	Thomas, David	1882
1873.	Tunner, Peter R. von	1897
1885.	Wedding, Hermann	1908

LIST OF THE MEETINGS OF THE INSTITUTE AND THEIR LOCALITIES FROM ITS ORGANIZATION TO OCTOBER, 1912.

No.	Place.	Date.	Vol.	Trans. Page.
1.	Wilkes-Barre, Pa.*	May, '71	1	3
2.	Bethlehem, Pa.	Aug., '71	1	10
3.	Troy, N. Y.	Nov., '71	1	13
4.	Philadelphia, Pa	Feb., '72	1	17
5.	New York, N. Y.*	May, '72	1	20
6.	Pittsburg, Pa	Oct., '72	1	25
7.	Boston, Mass	Feb., '73	1	28
8.	Philadelphia, Pa.*	May, '73	2	3
9.	Easton, Pa	Oct., '73	2	7
10.	New York, N. Y	Feb., '74	2	11
11.	St. Louis, Mo.*	May, '74	3	3
12.	Hazleton, Pa	Oct., '74	3	8
13.	New Haven, Conn	Feb., '75	3	15
14.	Dover, N. J.*	May, '75	4	3
15.	Cleveland, O	Oct., '75	4	9
16.	Washington, D. C	Feb., '76	4	18
17.	Philadelphia, Pa.†	June, '76	5	3
18.	Philadelphia, Pa	Oct., '76	5	19
19.	New York, N. Y	Feb., '77	5	27
20.	Wilkes-Barre, Pa.*	May, '77	6	3
21.	Amenia, N. Y	Oct., '77	6	10
22.	Philadelphia, Pa	Feb., '78	6	18
23.	Chattanooga, Tenn.*	May, '78	7	3
24.	Lake George, N. Y	Oct., '78	7	103
25.	Baltimore, Md.*	Feb., '79	7	217
26.	Pittsburg, Pa	May, '79	8	3
27.	Montreal, Canada	Sept., '79	8	121
28.	New York, N. Y.*	Feb., '80	8	275
29.	Lake Superior, Mich	Aug., '80	9	1
30.	Philadelphia, Pa.*	Feb., '81	9	275
31.	Staunton, Va	May, '81	10	1
32.	Harrisburg, Pa	Oct., '81	10	119
33.	Washington, D. C.*	Feb., '82	10	225
34.	Denver, Colo	Aug., '82	11	1
35.	Boston, Mass.*	Feb., '83	11	217
36.	Roanoke, Va	June, '83	12	3
37.	Troy, N. Y	Oct., '83	12	175
38.	Cincinnati, O.*	Feb., '84	12	447
39.	Chicago, Ill	May, '84	13	1
40.	Philadelphia, Pa	Sept., '84	13	285
41.	New York, N. Y.*	Feb., '85	13	585
42.	Chattanooga, Tenn	May, '85	14	1
43.	Halifax, N. S	Sept., '85	14	307
44.	Pittsburg, Pa.*	Feb., '86	14	587
45.	Bethlehem, Pa	May, '86	15	lxiii.
46.	St. Louis, Mo	Oct., '86	15	lxx.
47.	Scranton, Pa.*	Feb., '87	15	lxxvii.
48.	Utah and Montana	July, '87	16	xvii.
49.	Duluth, Minn	July, '87	16	xxiv.
50.	Boston, Mass.*	Feb., '88	16	xxviii.
51.	Birmingham, Ala	May, '88	17	xix.
52.	Buffalo, N. Y	Oct., '88	17	xxiv.
53.	New York, N. Y.*	Feb., '89	17	xxxi.
54.	Colorado	June, '89	18	xvii.
55.	Ottawa, Canada	Oct., '89	18	xxiv.
56.	Washington, D. C.*	Feb., '90	18	xxx.
57.	New York, N. Y	Sept., '90	19	vii.
58.	New York, N. Y.*	Feb., '91	19	xxv.
59.	Cleveland, O	June, '91	20	xvi.
60.	Glen Summit, Pa	Oct., '91	20	lxi.
61.	Baltimore, Md.*	Feb., '92	21	xix.
62.	Plattsburgh, N. Y	June, '92	21	xxxiii.
63.	Reading, Pa	Oct., '92	21	xliv.
64.	Montreal, Canada*	Feb., '93	21	lii.
65.	Chicago, Ill	Aug., '93	22	xiii.
66.	Virginia Beach, Va.*	Feb., '94	24	xvii.
67.	Bridgeport, Conn	Oct., '94	24	xxxv.
68.	Florida‡	Mar., '95	25	xix.
69.	Atlanta, Ga	Oct., '95	25	xxxiii.
70.	Pittsburg, Pa.*	Feb., '96	26	xvii.
71.	Colorado	Sept., '96	26	xxix.
72.	Chicago, Ill	Feb., '97	27	xvii.
73.	Lake Superior	July, '97	27	xxx.
74.	Atlantic City, N. J.*	Feb., '98	28	xvii.
75.	Buffalo, N. Y	Oct., '98	28	xxxvi.
76.	New York, N. Y.*	Feb., '99	29	xvii.
77.	California	Sept., '99	29	xlix.
78.	Washington, D. C.*	Feb., '00	30	xix.
79.	Canada	Aug., '00	30	xlv.
80.	Richmond, Va.*	Feb., '01	31	xix.
81.	Mexico	Nov., '01	32	cxviii.
82.	Philadelphia, Pa.§	May, '02	33	xxxv.
83.	New Haven, Conn	Oct., '02	33	xlvii.
84.	Albany, N. Y.*	Feb., '03	34	xxiii.
85.	New York, N. Y	Oct., '03	34	lxi.
86.	Atlantic City, N. J.*	Feb., '04	35	xxiii.
87.	Lake Superior	Sept., '04	35	xlii.
88.	Washington, D. C	May, '05	36	xlii.
89.	British Columbia	July, '05	36	liii.
90.	Bethlehem, Pa	Feb., '06	37	xli.
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102.	New York, N. Y	Feb., '12	43	lxxvii.
103.	Cleveland, Ohio	Oct., '12	44	.....

* Annual meeting for the election of officers. The rules were amended at the Chattanooga meeting, May, 1878, changing the annual election from May to February.

† Begun in May at Easton, Pa., for the election of officers, and adjourned to Philadelphia.

‡ Begun in February at New York City, for the election of officers, and adjourned to Florida.

§ " " " " " " " " " " " " to Philadelphia.

PUBLICATIONS.

THE publications of the Institute comprise:

TRANSACTIONS.

The volumes of *Transactions*, which are published annually, contain the list of officers, Constitution, etc., the Proceedings, and the papers revised for final publication. (In this revision, after the preliminary publication, authors are permitted to use the largest liberty; and the changes and additions made in papers are sometimes important. It should be borne in mind by those who study or quote a paper in the preliminary edition, that they may not have in that form the ultimate and deliberate expression of the author's views. It should be added, however, that in the majority of cases there are no important changes.) These volumes are for sale as follows, in paper covers:

Vols. I. to IV. (inclusive), each,	$3.00
Vols. V. to VIII. (inclusive), each,	4.00
Vol. IX.,	10.00
Vols. XI. to XXIX. (inclusive), each,	5.00
Vols. XXX. and XXXI., each,	6.00
Vol. XXXII.,	5.00
Vols. XXXIII. to XLII. (inclusive), each,	6.00

Half-morocco binding, $1 extra per volume.

The Board of Directors has authorized the following offers of sets of back volumes of the *Transactions*, at considerably reduced prices, to Members, Libraries, and Scientific Societies:

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I. Five volumes, bound in half-morocco, from No. 36 (1906) to No. 40 (1910),	$20
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BULLETIN.

Per annum, $10.00. (To members of the Institute, public libraries, educational institutions and technical societies, $5.00.

Single numbers, $1.00. (To members of the Institute, etc., $0.50.)

Index, Vols. I. to XXXV. (inclusive).—This volume, an octavo of 706 pages, affords a ready and complete reference to any subject treated or alluded to in the *Transactions*, Vols. I. to XXXV., inclusive. The names of persons, mines, works, towns, etc., have been included; and abundant cross-references and classified subheadings have been added to facilitate rapid consultation.

Bound in cloth, $5.00, half-morocco, ... $6.00

Index, Vols. XXXVI. to XL. (inclusive).—This work, containing in rearranged and condensed form the indexes of the *Transactions*, Vols. XXXVI. to XL., inclusive, is of special value in that it supplements the General Alphabetical and Analytical Index of Vols. I. to XXXV., and, by giving all of the new material contained in the technical and professional papers which have been contributed to the Institute during the five years, brings the index of all the volumes so far published fully up to the date of Vol. XL., June, 1910. An improvement has been made in the new arrangement of the material under group-headings, which, by presenting the references in tabular form, will enable the reader to find a given item more readily than in the former compact arrangement.

Bound in cloth, $1.50, half-morocco, ... $2.50

The Institute maintains at more than a hundred important mining centers throughout the world, free sets of its *Transactions*, open for consultation without fee, to all suitable applicants. Hence, the value of these indexes is by no means limited to individual possessors of complete sets of the *Transactions*. Moreover, the title of a paper, or the record of any remarks concerning a subject, being found in the Index, the Secretary's office of the Institute will supply upon written application any desired information as to the nature and length of said paper, whether it can be supplied in separate pamphlet form, etc.

SPECIAL EDITIONS.

"*The Genesis of Ore-Deposits*," comprising the famous treatise of the late Professor Franz Posepny, with the successive discussions thereof by Le Conte, Blake, Winchell, Church, Emmons, Becker, Cazin, Rickard, and Raymond (all of which were published in Volumes XXIII. and XXIV. of the *Transactions* of the Institute, and subsequently in the special "Posepny Volume," issued by the Institute); also, later papers by Van Hise, Emmons, Weed, Lindgren, Vogt, Kemp, Blake, Rickard, and others, and the discussions of these papers by De Launay, Beck, and many others (some of

these were included in Volume XXX. and the remainder appeared in Volume XXXI.); also a complete bibliography of Institute papers and discussions on this subject from 1871 to 1902.

The original Posepny volume comprised 265 pages, and was sold for $2.50, at which price the edition was long since exhausted. The present volume is an octavo of 825 pages.

Bound in cloth, ... $6.00

"*Ore-Deposits*," by Samuel Franklin Emmons, a continuation of the *Genesis of Ore-Deposits*, by Prof. Franz Posepny, comprising 29 papers descriptive of ore-deposits and discussions of their origin. Edited, with an introduction, by the late Dr. S. F. Emmons. The volume contains also a Biographical Notice of Dr. Emmons by his associate and friend, Dr. George F. Becker, and a comprehensive Bibliographical Index of the Science of Ore-Deposits, prepared by Prof. John D. Irving, H. D. Smith, and H. G. Ferguson. Dr. Emmons had finished his editorial work and written his Introduction before his lamented death in 1910; and the Volume contains his last words upon the subject to which he had given the work of his life, and on which he was justly regarded as the foremost authority. With the exception of a contribution by Mr. Penrose and the Bibliography, by Professor Irving, these papers have appeared in our *Transactions*, chiefly during the period 1900 to 1911. The authors are:—W. L. Austin, George J. Bancroft, George F. Becker, W. P. Blake, C. R. Boyd, John A. Church, Charles W. Dixon, J. R. Don, S. F. Emmons, W. H. Emmons, H. G. Ferguson, H. P. Gillette, J. B. Hastings, H. M. Howe, John D. Irving, Walter P. Jenney, James F. Kemp, Joseph Le Conte, C. K. Leith, W. Lindgren, Henry Louis, William H. Merritt, W. G. Miller, R. A. F. Penrose, R. W. Raymond, H. D. Smith, H. Sjögren, A. C. Spencer, J. E. Spurr, E. A. Stevens, H. S. Washington, Walter Harvey Weed, H. V. Winchell, Arthur Winslow.

About 1000 pages, illustrated.

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"*The Evolution of Mine-Surveying Instruments.*" This is a volume of about 400 pages, containing the original paper of Mr. Dunbar D. Scott on that subject (*Transactions*, XXVIII.), first published in 1898, together with later papers, continuing the same subject, and discussions thereof, by Hoskold, Lyman, Davis and many others.

Bound in cloth, ... $3.50

Year Book, containing List of Members, Constitution, etc., paper; to Members of the Institute, $0.50; to others, 1.00

Glossary of Mining and Metallurgical Terms (1881), cloth, . 1.00

Spanish-American Mining and Metallurgical Glossary, bound in leather, pocket-size, 96 pages, ... 0.75

Chart for the Solution of Kutter's Formula, on cloth, . . . 0.50

Pamphlets.

1. The Minutes of the Proceedings of each Meeting.

2. Such of the papers presented or read by title at each Meeting as are furnished by the authors and approved by the Council for full publication. These papers are published separately in pamphlet form, and are marked "subject to revision." Beyond the *Bulletin* edition, a small supply is retained to meet subsequent demand. The stock is nearly complete from 1880. These papers are for sale at the following prices:

No. of Pages.	Single Copies.	10 Copies.	20 Copies.
24 or less	$0.25	$2.00	$3.50
25 to 48	0.30	2.50	4.50
49 to 80	0.35	3.00	5.00
81 to 96	0.40	3.50	6.00
97 to 128	0.45	3.75	6.25
129 to 144	0.50	4.00	6.50
145 to 160	0.55	4.25	6.75
161 to 176	0.60	4.50	7.00

Authors' Edition of Pamphlets.

Extra copies of pamphlets, if ordered before the printing of the *Bulletin*, will be furnished to members of the Institute at special rates.

CERTIFICATE OF INCORPORATION.

[AMENDED NOV. 22, 1912.]

WE the undersigned, being all persons of full age and citizens of the United States, and a majority residents of the State of New York, desiring to form a corporation pursuant to the provisions of the Membership Corporations Law for the purpose of incorporating, as provided in Section 5 of Article I. of said law, the existing unincorporated association known as American Institute of Mining Engineers, do hereby make, acknowledge and file this Certificate for that purpose, and do Certify as follows:

I. That the American Institute of Mining Engineers is an unincorporated association organized and existing with the object of promoting the arts and sciences connected with the economic production of the useful minerals and metals and the welfare of those employed in these industries by means of meetings for social intercourse and the reading and discussion of professional papers, and to circulate by means of publications among its members and associates the information thus obtained.

II. That the persons duly appointed or designated to manage the affairs of said association are designated by the rules thereof Members of its Council; that the undersigned are all members of said Council as the same was constituted on the 29th day of December, 1904.

III. That on said last-mentioned date a regularly called meeting of said association was held at its office in the Borough of Manhattan, City of New York; that thirty days before such meeting notice of the intention to incorporate said association was given by mail to each member thereof whose residence or post-office address is known; that at said meeting the following resolutions were offered, seconded and duly adopted by the unanimous vote of all its members then present, to wit:

"*Resolved*, That it is the sense of the members and associates of the American Institute of Mining Engineers in general meeting assembled that it is desirable and necessary for the well being of said association and its members and for the furtherance of the objects for which the same has been formed, that said association incorporate under the Membership Corporations Law of the State of New York;

"*And Further Resolved*, That the Members of the Council of this Association, or a majority thereof, be and they hereby are authorized, in accordance with the provisions of Section 5 of Article I. of the Membership Corporations Law, to incorporate this association for the same purposes for which it has been organized and conducted, in the manner provided in Article II. of said law;

"*And Further Resolved*, That the name of said corporation as hereby adopted by this meeting shall be American Institute of Mining Engineers.

"*And Further Resolved*, That the said incorporators shall be named in the Certificate of Incorporation as directors of such corporation until its first annual meeting, and that such directors and their successors in office shall be and they hereby are authorized to enact and adopt a Constitution and By-Laws for the government of said corporation."

IV. That the name of the proposed corporation is American Institute of Mining Engineers.

V. That the purposes for which this corporation is to be formed are: To promote the arts and sciences connected with the economic production of the useful minerals and metals and the welfare of those employed in these industries by means of meetings for social intercourse, and the reading and discussion of pro-

fessional papers, and to circulate by means of publications among its members the information thus obtained; and to establish and maintain a place of meeting for its members and a hall for the reading of papers and delivery of addresses, and a library of books relating to subjects cognate to the sciences and arts of mining and metallurgy.

VI. That the territory in which the operations of this corporation are to be principally conducted is the United States of America, and the principal office for the transaction of its affairs is to be located in the City, County and State of New York.

VII. That the duration of this corporation is to be perpetual.

VIII. That the number of Directors of this corporation is to be twenty-four, consisting of one President, two Past Presidents, six Vice-Presidents, and fifteen other Directors; that at the annual meeting to be held in February, 1913, there shall be elected to serve, jointly with the six members of the former Board whose terms will not then have expired, one President, who shall serve as such for one year, and as Past President for two years; two Past Presidents to be selected from former Presidents of the Institute, one to serve for one year and one to serve for two years; six Vice-Presidents, of whom two shall be elected to serve for one year, two for two years and two for three years, and nine Directors, of whom two shall be elected to serve for one year, two for two years and five for three years.

After the year 1913 there shall be elected at each annual meeting one President to serve as such for one year and as Past President for two years; two Vice-Presidents to serve for three years and five Directors to serve for three years.

At each subsequent annual meeting so many Directors only shall be elected as shall be necessary to fill vacancies then existing in the Board, and each Director then chosen shall be elected to serve for three years unless the vacancy which he shall be chosen to fill has been caused otherwise than by expiration of the former incumbent's term of office, in which event he shall be elected to serve for the balance of his predecessor's unexpired term.

IX. That the time for holding the annual meeting of this corporation shall be the third Tuesday of February in each year.

X. That the names and post-office addresses of the subscribers hereto who are to be directors of this corporation until the first annual meeting are

Names.	Post-Office Addresses.
James Gayley,	71 Broadway, New York City;
Frank Lyman,	88 Wall Street, New York City;
James F. Kemp,	Columbia University, New York City;
Charles H. Snow,	New York University, New York City;
Frank Klepetko,	24 State Street, New York City;
Thomas A. Rickard,	261 Broadway, New York City;
James Douglas,	99 John Street, New York City;
Albert R. Ledoux,	99 John Street, New York City;
Rossiter W. Raymond,	99 John Street, New York City.

The above certificate, dated December 30, 1904, was signed and acknowledged by the subscribers before William A. Lockwood, Notary Public (No. 74), New York County.

I, the undersigned, Justice of the Supreme Court of the State of New York, do hereby approve of the above Certificate of Incorporation of the American Institute of Mining Engineers, and do authorize the same to be filed.

SAMUEL GREENBAUM,
Justice of the Supreme Court.

January 7, 1905.

CONSTITUTION.

[ADOPTED FEB. 18, 1913.]

ARTICLE I.

NAME AND OBJECT.

SEC. 1. This Institute is incorporated under the Membership Corporations law of the State of New York; its corporate name is American Institute of Mining Engineers; and its objects are such as are stated in its Certificate of Incorporation.

ARTICLE II.

MEMBERS.

SEC. 1. The membership of the Institute shall comprise four classes, namely: 1. Members; 2. Honorary Members; 3. Associates; 4. Junior Members.

All members shall be equally entitled to the privileges of membership, excepting that Honorary Members, Junior Members, and Members and Associates whose residences shall be outside of the United States, Mexico, and Canada shall not be entitled to vote. Members and Associates residing within the United States of America, Mexico, and Canada, and not in arrears for dues, shall be entitled to vote in person at the meetings of the Institute, or, as hereinafter provided for, by letter ballot.

SEC. 2. The following classes of persons shall be eligible for membership in the Institute, namely: as Members, all professional mining engineers, geologists, metallurgists, or chemists, and all persons actively engaged in mining and metallurgical engineering, geology, or chemistry; as Associates, all persons desirous of being connected with the Institute who in the opinion of the Board of Directors are suitable.

As Junior Members, all students in good standing in engineering schools who have not taken their degrees and who are nominated by at least two of their instructors. A Junior Member may remain such not longer than three years after leaving the engineering school, at the end of which period his qualifications to become a Member or Associate must be passed upon by the Membership Committee. If elected he shall pay at that time the entrance fee and dues of a Member or Associate.

Every candidate for election as a Member, Associate, or Junior Member must be proposed for election by at least three Members or Associates, must be approved by the Committee on Membership, as prescribed in the By-Laws, and must be elected by the Board of Directors.

SEC. 3. Not less than three-fourths of the votes cast shall be necessary to an election; newly elected candidates shall be immediately notified of their election and also of their obligations, by the Secretary of the Institute. Every person so elected shall become a Member or Associate or Junior Member of the Institute in

the class to which he is chosen, upon payment of his initiation fee, if required by this Constitution, and first year's dues, as hereinafter stated. Each candidate for Honorary Membership must be recommended by at least ten Members and must be elected by ballot at a meeting of the Board of Directors by the unanimous vote of the Directors present; provided, however, that the number of Honorary Members shall at no time exceed twenty.

SEC. 4. If any person elected a Member, Associate, or Junior Member does not within four months after the mailing of notice of his election accept the same and pay his initiation fee, if required, and dues for the year in which he was elected, his election shall be null and void and he shall be notified thereof by the Secretary of the Institute; providing, however, that if elected in the months of October, November, or December, the candidate may, at his option, pay dues only for the ensuing year, in which case he shall be entitled to receive the *Bulletin* of the Institute for the months of November and December, but not the annual volume of the year during which he was elected to membership.

SEC. 5. The Board may at any time, for good and sufficient reasons, change the classification of a member of any class except Honorary Members.

ARTICLE III.

DUES.

SEC. 1. The dues of Members and Associates shall be, until otherwise determined by the Institute, ten dollars per annum, and those of Junior Members shall be five dollars per annum. All dues shall be payable in advance on the first day of each calendar year, and notice to this effect shall be published in the last number of the *Bulletin* of each calendar year, and a bill shall be mailed to each Member, Associate, and Junior Member on the first day of the month of January in each year, stating the amount of dues and date when payable and the penalty and conditions incident to default in payment within the limit of time set by the Constitution.

Each newly elected Member or Associate shall pay, immediately on notification of his election, an initiation fee of ten dollars; Junior Members shall not pay any initiation fee. Honorary Members shall not be liable for initiation fees or dues.

If any Member, Junior Member, or Associate is in arrears for four months, the *Bulletin* shall no longer be sent to him and he shall be notified by the Secretary of the Institute that no publications will be sent him until his arrears be paid. Any Member, Junior Member, or Associate in arrears for one year shall be dropped from the rolls unless the time for payment is extended by the Directors. A member so dropped may be reinstated by the Directors on paying all arrears of dues.

SEC. 2. The Board of Directors shall have the right to increase the dues of the membership for the next ensuing year or years by any sum not to exceed the amount of five dollars per annum, such increase of dues to be applicable solely to the extinction of the principal of the land debt of the Institute. As soon as such indebtedness is paid the dues shall automatically return to those provided in Section 1 of this Article.

SEC. 3. Any Member or Associate not in arrears may become, by the payment of one hundred and fifty dollars at one time, a Life Member or Life Associate, and shall not be liable thereafter to pay annual dues. The money thus received shall be invested and only the income thereof used for current expenses of the Institute.

ARTICLE IV.

Business Meetings of the Institute.

Sec. 1. The annual meeting of the Institute for the election of Directors and transaction of other business shall be held at the headquarters of the Institute in the City of New York on the third Tuesday in February in each year. A report of the financial condition of the Institute and an abstract of the accounts shall be furnished by the Directors, and presented at each annual meeting.

Sec. 2. Special business meetings of the Institute may be held at such times and places in the State of New York as the Board of Directors may appoint, upon notice to all Members and Associates entitled to vote, directed to each at his last known post-office address, and mailed not less than twenty days before the date fixed for such meeting.

Sec. 3. At all business meetings of the Institute the presence of twenty-five Members and (or) Associates shall constitute a quorum.

Sec. 4. At all business meetings of the Institute, Members and Associates entitled to vote may vote only in person. Any matter which is presented over the signatures of twenty-five Members or Associates, or which is advocated or opposed by twenty-five Members or Associates present at a business meeting of the Institute, or which the Board of Directors may wish to refer to the membership for decision, shall be sent out to the membership for a vote thereon by letter ballot, accompanied by a statement prepared by the Board of Directors giving the arguments for and against its adoption. Only Members and Associates entitled to vote may vote thereon.

Sec. 5. All meetings of the Institute other than business meetings shall be held at such times and places as the Board may appoint. Notice of all such meetings shall be given to all Members, Associates, and Junior Members by mail.

ARTICLE V.

Directors and Officers.

Sec. 1. The business, financial, professional, technical, scientific and social interests of the Institute shall be managed by and under a Board of Directors, elected annually by letter ballot in the manner specified in this Constitution or in the By-Laws, to serve for the terms set forth in the Certificate of Incorporation.

Sec. 2. The Board of Directors shall consist of a President, two Past Presidents, six Vice-Presidents, and fifteen Directors.

Eight Directors shall be elected annually by letter ballot: One of said eight shall be designated as "Director and President" and two shall be designated as "Director and Vice-President."

Sec. 3. The officers of the corporation, who shall also be termed the respective officers of the Institute, shall be a President, First Vice-President, Secretary, and Treasurer. The President shall be that Director who is chosen at the annual election as "President." He shall serve as President for one year, and for the subsequent two years of his directorate be termed "Past President of the Institute." The President shall be ineligible for re-election as such until two years after expiration of his term as President.

At the first meeting of the Board of Directors after the annual business meeting of the Institute, the Board shall elect a First Vice-President, to hold office for one year, from among the Vice-Presidents of the Board; a Secretary, who may or may not be a member of the Board, to serve for one year; and a Treasurer from among

the members of the Board, to hold office for one year. The Secretary or the Treasurer may be removed from office at any time by a unanimous vote of the Directors present at a special or regular meeting of the Board, due notice having been given in advance that such action is to be considered at said meeting. The Secretary shall receive such salary as may be fixed by the Board of Directors.

The duties of all officers shall be such as usually pertain to their respective offices, together with such other duties as may from time to time be prescribed for them by the By-Laws or Board of Directors. The Treasurer shall give a bond for the faithful performance of his duties, in a sum to be fixed by the Board of Directors, but at the expense of the Institute.

SEC. 4. In the event of a vacancy occurring in the Board of Directors by death, resignation, promotion by election as President or Vice-President, or for any other reason than retirement at the end of three years' service, the remaining members of the Board shall elect a successor to fill the vacancy and to serve for the unexpired term. A member of the Board whose term has not expired and who is elected President or Vice-President at an annual election shall be considered to have vacated the former office held by him, and the Board shall fill the vacancy as above provided.

SEC. 5. The Board of Directors may, from time to time, in its discretion, appoint any Past Secretary of the Institute "*Secretary Emeritus*," to receive such compensation and to perform such duties as the Board may designate.

SEC. 6. The Board of Directors may authorize the formation of Local Sections, Technical Committees, or arrange affiliations with other Societies in accordance with this Constitution and the By-Laws.

SEC. 7. The Board of Directors may appoint standing and special committees charged with such duties as the Board may assign. Such committees shall be directly responsible to, and report to the Board, and take only such responsibilities as the Board shall authorize.

SEC. 8. At the first meeting of the Board of Directors after the annual meeting of the Institute, it shall appoint an Executive Committee of five of its members, three of whom shall constitute a quorum. This committee may act for the Board between meetings of the Board and at any regular or special meeting whenever a quorum of six Directors is not present. The Executive Committee is not authorized to perform the following acts, or any others the Board may see fit to interdict, viz. : fill vacancies on the Board ; elect the First Vice-President, Secretary, or Treasurer ; fix salaries of officers (it may fix those of employees) ; vote extra compensation to officers ; vote extraordinary expenses or incur obligations of over five hundred dollars ; take action affecting already invested funds or property rights of the Institute ; elect Honorary Members ; expel Members.

SEC. 9. The Chairmen of Local Sections or appointed representatives thereof, and the chairmen of the Technical Committees or Divisions of the Institute, if not members of the Board of Directors, shall be notified of and have the privilege of attending all meetings of the Board, but without the right to vote. They may participate in discussion, and shall receive the minutes of the Board's meetings.

SEC. 10. All salaries paid by the Institute shall be fixed in advance by the Board of Directors, but no salary shall be fixed in advance for a period longer than twelve months.

SEC. 11. The funds of the Institute shall be paid out only by check signed by the Treasurer and countersigned by the Secretary ; in case of absence or disability, the President or First Vice-President may act for the Treasurer, or any Vice-President for the Secretary.

SEC. 12. The Directors may appoint an Assistant Secretary, who shall, under the instructions of the Board, have charge of such of the business affairs of the Institute as may be assigned to him by the Board. His salary shall be fixed by the Board, and he shall hold office during its pleasure.

ARTICLE VI.

MEETINGS OF THE BOARD OF DIRECTORS.

SEC. 1. A regular meeting of the Board of Directors for the election of officers and the transaction of other business shall be held in February of each year, at the headquarters of the Institute, in New York City, after the annual business meeting of the Institute has adjourned.

SEC. 2. The Board of Directors shall also meet once each month except in July and August, on a regular date to be fixed at its February meeting, for the transaction of such business as may come before it.

A conference of the Board may be called at the request of three members of the Board, or by the President, to meet at any time or place, in connection with a meeting of the Institute for reading and discussion of papers or for the discussion of matters affecting the welfare of the Institute. All recommendations adopted by a majority of those present at such conference shall be transmitted to the Secretary of the Institute, accompanied by a record of the number favoring such recommendations, together with notes of the discussion thereon, and by him brought before the next meeting of the Board of Directors for consideration and such action as it may deem expedient.

Special meetings of the Board shall be called by the Secretary at the request of the President, First Vice-President, or any three members of the Board, to meet at any time and place, by notice mailed to the members of the Board at least ten days in advance, or by telegram sent at least five days in advance of the date of the meeting to members in North America (Alaska excepted).

SEC. 3. At all meetings of the Board of Directors the presence of six members shall constitute a quorum. In the absence of said quorum at any meeting of the Board, a quorum of the Executive Committee, if present, shall transact such necessary business as is within the powers delegated to that Committee.

ARTICLE VII.

NOMINATION AND ELECTION OF DIRECTORS AND OFFICERS.

SEC. 1. A ticket of nomination for offices and places annually falling vacant in the Board of Directors shall be prepared by a Committee on Nominations, of which no member shall be, at the time, on the Board. This Committee shall consist of the Chairmen of three Local Sections, one ex-President, and three other Members of the Institute, and, on recommendations made by the President of the Institute, shall be appointed by a vote of the Board of Directors, within ninety days after the annual meeting, and it shall then proceed to the selection of candidates and the naming of a ticket. In making such selections, the nominating committee shall, so far as practicable, distribute the representation on the Board geographically, so that seven members shall be residents of the district including New York City and the territory within a radius of fifty miles of the headquarters of the Institute, and one member a resident of each of the geographical districts enumerated in the By-Laws.

The official ticket thus formulated shall be transmitted to the Secretary of the Institute, not later than the last Friday in October, be submitted to the Board at

its October meeting for its consideration, published in the November *Bulletin*, and not later than January 1 be printed and sent to the membership. Any complete or partial ticket of nominees signed by any twenty-five Members or Associates of the Institute and transmitted to the Secretary by December 15 shall also be printed and circulated with the official ticket and over the names and with the recommendations of the nominators, together with an expression of opinion of the Board thereon if deemed expedient.

SEC. 2. The election ballots mailed with the nominations shall contain space for the naming of eight Directors, of which the first shall be designated: "For Director and President," the second and third: "For Director and Vice-President," and the rest: "For Director." They shall be accompanied by small inside adhesive envelopes marked: "Ballot. Do not sign," and outer mailing envelopes addressed to the Secretary and marked "Enclosing Ballots only." Members voting shall not sign their names to the ballot, but seal it in the inner envelope, enclose it in the outer mailing envelope, and write their names on the outside, in the place provided. Cumulative voting shall not be permitted. Letter ballots must reach the Secretary on or before the second Tuesday in February. He shall give them unopened to tellers, to be appointed by the Board of Directors at its January meeting, at least three days before the annual business meeting, marking plainly those from Members not entitled to vote, because of geographical location or by being one year or more in arrears for dues. The tellers shall open and count the valid ballots received from Members entitled to vote, rejecting any unidentified by the name of the sender on the outer envelope; they shall prepare a report of the voting and hand it to the Chairman of the annual business meeting of the Institute. The latter shall declare elected to the respective offices the eligible persons receiving the highest number of votes. The ballots and unopened letters shall be returned to the Secretary at the annual business meeting; the latter shall preserve them one month, and then destroy the ballots which have been counted, and open and destroy the others.

SEC. 3. At the annual election in February, 1913, at which time the former Board of nine Directors, of whom three will on that day retire, is, under the amended Certificate of Incorporation, to be succeeded by a new Board of twenty-four Directors, the Institute shall elect by letter ballot in the manner hereinabove provided (save that nomination therefor shall have been made by a committee appointed for that purpose by the Board of Directors prior to December 15, 1912), the following new Directors:

One President and Director, who shall serve as President for one year and as Past President for two years; two Past Presidents to be selected from former Presidents of the Institute, one to serve for one year and one to serve for two years; six Vice-Presidents, of whom two shall be elected to serve for one year, two for two years, and two for three years; and nine Directors, of whom two shall be elected to serve for one year, two for two years, and five for three years.

ARTICLE VIII.

PAPERS AND PUBLICATIONS.

SEC. 1. The Board of Directors shall appoint a Committee on Papers and Publications, of which the Secretary shall be a member, whose duty shall be to pass upon the fitness of technical communications submitted for publication by the Institute. Stenographic or contributed discussions of papers may be passed upon by the Secretary, in consultation with members of the above committee.

SEC. 2. The Institute shall not assume responsibilities for any statements of fact or opinion advanced in the papers or discussions at its meetings. Neither the Board nor the Institute shall officially approve or disapprove any technical or scientific opinion, or of any proposed enterprise which is outside of the management of the meetings, discussions and publications of the Institute, and the conduct of its business affairs by the Board of Directors.

SEC. 3. Special committees may from time to time be appointed by the Board to make investigations and prepare reports for presentation to the Institute, but no action shall be taken binding the Institute for or against the conclusions embodied in any such reports.

ARTICLE IX.

SUSPENSIONS AND EXPULSIONS.

SEC. 1. Any member of the Institute who shall be convicted of a crime involving, in the opinion of the Board of Directors, moral turpitude, shall, upon the passage by the Board of Directors of a resolution declaring the crime for which he has been convicted to be of such character, be thereupon dropped from membership in this Institute.

SEC. 2. Any Member of the Institute may be suspended or expelled for misconduct by the Board of Directors, after charges setting forth such misconduct shall have been prepared and filed in writing with the Board. Upon the receipt of such charges in writing, the Board may, in its discretion, suspend such Member pending a hearing and determination thereupon. As soon as may be after the receipt of such charges, the Board shall fix a date for a hearing thereupon and shall give to the accused Member notice thereof in writing, mailed to him at his last known post office address not less than thirty days before said date, accompanied by a copy of the charges and a copy of the second, third and fourth sections of this article.

SEC. 3. Upon the day fixed for the hearing, the accused Member may appear before the Board, either in person or by an accredited representative; hear any witnesses who may be called in support of the charges and, at his option, cross-examine the same, and hear read any documentary evidence offered in support of the charges. The accused may, in his discretion, produce and examine witnesses in his defense, and submit documentary evidence, including a statement from himself in writing. After the conclusion of the hearing, the Board of Directors shall consider and vote to approve or disapprove the charges. If the Board shall, by a vote of two-thirds of its members, declare the charges sustained, it may suspend the Member for a stated period, or expel him.

SEC. 4. If the accused Member shall not appear at the hearing, and shall within three months thereafter file with the Board an affidavit stating that he had not received notice of the charges against him in time to enable him to present his defense, the Board shall fix a date for a re-hearing within three months from the receipt of such affidavit, and shall immediately notify the accused Member by mail of such date. Upon the re-hearing, the accused shall have the same privilege of presenting his defense as he would have had upon the original hearing; and after the defense is presented, the Board shall take a new vote upon the charges, the result of which shall be conclusive.

ARTICLE X.

AMENDMENTS.

SEC. 1. Proposals to amend this Constitution shall be presented in writing to the Board of Directors, signed by at least ten Members; they shall reach the Secretary not later than October fifteenth. The Board of Directors shall consider them and the proposers shall be notified of the opinion of the Board in regard to them, and they may then either withdraw their proposals, or accept any changes suggested, or insist on the original form. The original form, if insisted upon, together with the changes proposed by the Board, if such there be, shall be mailed to all Members on or before January first, with the election ballots, and with a ballot in proper form for voting upon the amendment in its original or amended form. Ballots shall be received by the Secretary until the second Tuesday in February. The Tellers of the election shall count these ballots, and report the result to the Chairman of the annual meeting. Any members present in person at the annual meeting who have not voted on the amendments by ballot, shall have the privilege of having their votes recorded before the result of the voting is announced. Any proposed amendment which shall have been accepted by a majority of the votes thus cast, shall be adopted.

BY-LAWS.

I. Presiding Officer.

At all business meetings of the Institute the President, or, in his absence, the First Vice-President, or, in the absence of both of them, any other Vice-President or Director, chosen by the meeting, shall preside.

At all other meetings of the Institute the President, or, in his absence, one of the Vice-Presidents, shall preside. If none of these be present the meeting shall elect a Chairman.

II. Order of Business.

At each business meeting of the Institute the order of business shall be as follows:

1. Reading of minutes of preceding meeting.
2. Report of the President.
3. Report of the Treasurer.
4. Report of the Secretary.
5. Election of Directors.
6. Reports of Standing Committees.
7. Reports of Special Committees.
8. Special Orders.
9. Miscellaneous business.

This order of business may be changed at any meeting, by a vote of a majority of the Members and Associates present.

At all sessions of the Institute, other than business meetings, the order of proceedings and the time of adjournment shall rest in the discretion of the presiding officer.

III. Secretary of the Institute.

The Secretary shall keep a record of the proceedings of all meetings of the Institute. He shall be custodian of the corporate seal, of the minute books, and of all legal documents belonging to the Institute. He shall have charge, on behalf of the Institute, of all correspondence except such as pertains directly to the office of the Treasurer.

He shall notify all Officers and Directors and all members of committees of their election and appointment; shall issue notices of all meetings of the Board, and of the annual and other business meetings of the Institute; and shall, in calling special meetings of the Directors, specify the objects of such meetings. He shall act as clerk at all the meetings of the Board of Directors and at all meetings of the Institute.

The Secretary shall be custodian of all technical or scientific papers submitted to the Institute for its consideration, shall have general charge and supervision of the editing and proof-reading of all material published by the Institute, and of the distribution thereof, and be *ex officio* Chairman of the Committee on Publications.

On the first day of May following the year in which each volume of *Transac-*

tions is printed, he shall turn over to the Library Committee all copies of the same not theretofore distributed by him.

The Secretary may, with the approval of the Board of Directors, employ such persons as are necessary to constitute a clerical and office force, at such salaries as shall be approved by the Board of Directors or its Executive Committee. He shall be the immediate superior of all such employees.

IV. Assistant Secretary.

The Board of Directors may, upon request of the Secretary, appoint an Assistant Secretary, to take charge of such portion of the Secretary's duties as may be assigned him by the Board or its Executive Committee, such as supervising the printing and distribution of advance papers, of the *Bulletin* and *Transactions*, soliciting advertising and conducting general business correspondence.

During the absence of the Secretary at meetings of the Institute, or for other reasons, he shall act as Secretary *pro tem.*

V. Treasurer.

The Treasurer shall collect and, under the direction of the Board of Directors, shall disburse all funds of the Institute. He shall keep regular accounts in books belonging to the Institute, which shall be open to any member of the Board of Directors. He shall report in writing at each annual meeting of the Institute, and at every regular meeting of the Board of Directors, the balance of money on hand, and any existing appropriations which may affect the same.

His accounts shall be audited annually by a committee of three Members or Associates to be appointed by the President at least thirty days prior to the annual meeting in each year, which committee shall report thereon at such annual meeting. The committee may employ a public accountant to assist it, at the expense of the Institute.

The Treasurer may, at his discretion, place funds of the Institute, not at any time exceeding five hundred dollars, in the hands of the Secretary, Assistant Secretary or Assistant Treasurer, and may delegate to either of them the duty of paying, therefrom, the smaller current expenses of the Institute. The Treasurer shall examine this "Cash Box" account each month.

The Treasurer shall be solely responsible to the Institute for all moneys received, whether the same are entrusted to others or not, and may require from any one to whom he entrusts funds, a bond, running to the Treasurer personally and taken at the expense of the Institute.

VI. Assistant Treasurer.

The Board of Directors may appoint an Assistant Treasurer, to whom it may delegate the duties of conducting, under the supervision of the Treasurer, correspondence incidental to the office of Treasurer, of collecting, receiving and depositing in bank, to the credit of the Institute, all moneys, and of paying, out of a special account, if entrusted to him, the smaller current expenses of the Institute, subject to the instructions of the Treasurer. He shall hold no other office.

VII. Standing Committees.

The standing committees of the Institute shall be four: A Finance Committee, a Library Committee, a Committee on Publications, and a Committee on Membership.

The Finance Committee shall consist of three members of the Board of Directors, who shall be appointed by the Board at its first meeting after the annual meeting of the Institute in each year.

The Library Committee shall consist of the Secretary of the Institute and four other Members or Associates of the Institute, one being appointed annually by the President, at the first meeting of the Board after the annual business meeting of the Institute, to serve for four years, in conformity with the By-Laws of the United Engineering Society, approved by the Board of Directors of the Institute.

The Committee on Publications shall consist of the Secretary of the Institute, who shall be its chairman, and at least twelve specialists, Members of the Institute, to assist in passing on all papers offered for publication. They shall be appointed annually by the newly-elected President.

The Committee on Membership shall consist of five Members of the Institute, the Chairman and at least one other member being chosen from the Board of Directors, and shall be appointed by the President at the first meeting of the Board after the annual business meeting of the Institute.

VIII. Finance Committee.

It shall be the duty of the Finance Committee to inquire into and examine the financial condition of the Institute and to consider proper means of increasing its revenues and of limiting its expenses. It shall report at each monthly meeting of the Board, and at other times whenever it shall desire or be directed so to do; and the Treasurer shall at all times furnish it with such statements and information as it may desire.

It shall determine, with the approval of the Board, the investment of such surplus moneys as shall from time to time accrue to the Institute. It shall, at least once in each year, examine the securities belonging to the Institute, and report thereon to the Board.

It may, at any time, examine the books and vouchers of the Treasurer and Assistant Treasurer.

The Treasurer shall not be a member of the Finance Committee, but shall attend the meetings of the same if requested.

The Finance Committee shall present a budget to the Directors at the meeting preceding the annual meeting of the Institute. This budget shall give an itemized estimate of the receipts and expenses of the Institute for the ensuing year. The Directors shall pass on this budget at their first meeting after the annual business meeting of the Institute, modifying it as they consider necessary, and make definite, detailed appropriations for the following year.

All bills, accounts, salaries, pay-rolls and claims of every kind against the Institute shall, before being paid, be examined by the Finance Committee and be approved by at least one member of the Committee. The Committee shall not authorize any payments in excess of appropriations. If at any time any of the appropriations seems in danger of being insufficient the Finance Committee shall report the same to the Directors.

IX. Library Committee.

The Library Committee shall constitute the official representatives of the Institute upon the Library Board of the United Engineering Society, and be guided by the By-Laws of that Society, as approved by the Board of Directors of this Institute, with full powers to act for the Institute within the appropriations allowed by the Board of Directors.

The Library Committee shall be the custodian of all books in the Institute library and of additions thereto; also of all back numbers of the *Transactions* of the Institute. It shall, on the first day of May of each year, receive from the Secretary and receipt for all the volumes of *Transactions* and *Bulletins* or other publications for the preceding year not previously distributed by said Secretary.

It shall cause to be kept, under the directions of the Secretary, a catalogue of all books in the library and an account in ledger form of all volumes of *Transactions* in its custody, in which shall be charged all volumes delivered to it, and in which shall be credited all volumes taken from its custody for sale or for any other purpose.

It shall make a written report annually to the Board of Directors, stating the stock of publications on hand, and a detailed statement of sales during the year, together with such observations and recommendations as it may wish, regarding said publications or the conduct of the library.

X. Committee on Publications.

1. This Committee shall perform its functions as follows:

(*a*) On the receipt of a paper by the Secretary (Chairman), he shall send it to the member of this Committee who, in his judgment, is most competent and available to pass upon it, as "Reader," accompanying the paper with his own opinion of its suitableness for publication, and any other pertinent information.

(*b*) If the Reader and the Chairman agree upon the suitability or unsuitability of the paper, it shall be considered accepted for publication or rejected, as the case may be.

(*c*) If these two do not agree, the paper shall be submitted to a third member of the committee, and the opinion of two of these three members shall decide the matter.

(*d*) If a paper has been refused publication, the author shall have the right of appeal, in which case the persons previously passing on the paper, together with others selected from the Committee by the President—making five altogether—shall decide the question.

(*e*) If a paper has been accepted for publication, it shall be considered eligible to be placed on the program of a meeting.

2. The placing of a paper upon the program of a meeting shall not give it the right to be published in the *Bulletin* or *Transactions* of the Institute; its suitability for publication must in every case be passed upon by the Committee, as provided for in Section 1 of this Article.

3. In case the Secretary is unable to secure a decision as to the suitability or unsuitability of a paper for publication, as directed in Section 1, before the time of announcing the program of a meeting, he may at his own discretion place the paper upon the program of the meeting, or refuse it a place thereon.

XI. Committee on Membership.

All nominations for Members, Associates, or Junior Members of the Institute shall be submitted to and passed upon by the Committee on Membership. This Committee shall meet at least once in each calendar month, with the exception of the months of July and August. It shall receive and consider all communications respecting candidates, and shall make diligent inquiry as to the character and qualification of each. Its proceedings shall be reported to the Board, if required, but otherwise shall be secret and confidential.

No member of the committee shall propose any candidate.

XII. Election of Members.

After the Committee on Membership shall have reported to the Board its conclusions as to the acceptability of each candidate, the Board shall vote upon the same. No person shall be proposed for election to the Institute within one year after his name shall have been rejected by the Board.

XIII. Nominations.

The geographical districts to be considered by the Committee on Nominations shall be as follows, until otherwise ordered by the Board.

District No. 1. New England, New York, and New Jersey, excepting New York City and district, which is provided for in the Constitution.

District No. 2. Pennsylvania.

District No. 3. Ohio, Indiana, Illinois, Iowa, and Missouri.

District No. 4. Minnesota, Wisconsin, and Michigan.

District No. 5. Montana, North and South Dakota, Wyoming, Nebraska, Kansas, Washington, Oregon, Idaho, and Alaska.

District No. 6. California and Nevada.

District No. 7. Utah, Colorado, Arizona, and New Mexico.

District No. 8. Louisiana and Texas.

District No. 9. Other Southern States and District of Columbia.

District No. 10. Mexico.

District No. 11. Canada.

XIV. United Engineering Society.

This Institute shall be represented upon the Board of Trustees of the United Engineering Society by three Members or Associates, one retiring each year, as provided in the By-Laws of the said United Engineering Society.

At the December meeting of the Board of Directors the Board shall designate a Member or Associate of this Institute to be a representative of this Institute upon the Board of Trustees of the said United Engineering Society for a period of three years beginning at the next ensuing annual meeting of said Society.

At any time when a vacancy shall occur in the representation of this Institute on the Board of Trustees of said Society, the Board of Directors of this Institute shall designate a Member or Associate to fill such unexpired term.

These representatives shall report in writing to the Board of Directors at its regular meetings held in March, June, September, and December, and at any other meeting, if called upon by the Board.

XV. Publication of a Bulletin.

The publications of the Institute shall include, in addition to the *Transactions*, a periodical called the *Bulletin* of the American Institute of Mining Engineers which shall contain reports of proceedings, professional papers, notices and other matter of interest to Members. From the annual dues paid by each Member, Associate, and Junior Member, five dollars shall be deducted and applied as a subscription to the *Bulletin* for the year covered by such payment. Special volumes upon technical subjects may be issued, by vote of the Board.

XVI. Obligations to Members.

Any Member of whatever class whose dues are paid within the limits set by the Constitution shall receive the *Bulletin;* and all Members and Associates who pay

ten dollars annual dues, also all Life and Honorary Members, shall receive in addition the *Transactions.*

XVII. Divisions.

Sec. 1. Professional groups to be known as Divisions of the Institute and to be organized from its members, may be authorized by the Board of Directors. Any Member of the Institute may register for membership in any Division in which he is interested and may resign therefrom provided he be under no financial obligation thereto.

Sec. 2. The officers of the Division shall be a Chairman, one or more Vice-Chairmen, a Secretary-Treasurer, and an Executive Committee. The Chairman, Vice-Chairman, and Secretary shall be *ex officiis* members of the Executive Committee of the Division. All officers of Divisions shall be elected annually by letter ballot by the members of the Divisions, and shall take office at the close of the meeting at which they are elected. They shall hold office for one year or until their successors are elected.

Sec. 3. A Division shall have the right to make rules for its own government, subject to the approval of the Board of Directors, not inconsistent with the Constitution and By-Laws of the Institute. The Board of Directors shall have the right to amend, annul, or add to these rules.

Sec. 4. Any Division may raise or collect funds to be expended for its own purposes and may have the entire management and control of such funds in so far as said management or control does not conflict with the provisions of the Constitution and By-Laws of the Institute.

Sec. 5. The Board of Directors shall have the right to recommend the dissolution of any Division subject to approval of such recommendation at any annual meeting of the Institute, in which case the net property of such Division shall be divided pro rata among all members thereof in good standing.

Sec. 6. Each Division shall furnish the Board of Directors on the 1st of February in each year an annual financial statement of its affairs in such form as the Board of Directors may require.

XVIII. Local Sections.

Sec. 1. A Local Section of the Institute may be authorized by the Board at the written request of ten Members residing within an appropriate distance of a central point.

Sec. 2. The Board shall define the territory of a Section.

Sec. 3. Only one Section shall be authorized in one locality or district.

Sec. 4. A section must consist of at least twenty-five members; if its membership falls below twenty-five in number, the Board may annul the Section.

Sec. 5. Only Members and Associates of the Institute shall be members of its Local Sections.

Sec. 6. All Members and Associates of the Institute residing within the territory of a Section shall be eligible for membership in such Section. But any such person failing within three months, after due invitation, to become a member of such Local Section, shall thereafter be admitted to its membership and privileges only on such conditions as said Local Section shall determine.

Sec. 7. The officers of a Section shall be elected, after the formation of the Section has been duly authorized, at a meeting of the members of the Institute within the territory of said Section, called by the sponsors of the Section, notice of said meeting and its objects being given to said Members at least thirty days in advance. Officers of Local Sections shall be elected for a term not longer than one year.

SEC. 8. The Officers of a Local Section shall be a Chairman, Vice-Chairman, Secretary, Treasurer (or Secretary-Treasurer), and such others as the Section may desire.

SEC. 9. It shall be the policy of the Board of Directors of the Institute to contribute from its funds for the necessary running expenses of each Local Section, when and so far as practicable, an amount not exceeding, in each year, twenty-five per cent. of the dues received from the members of said Section in said year, but in no case exceeding the sum of two hundred and fifty dollars. Requests for such appropriation shall be signed by the Chairman, Secretary and Treasurer of the Section.

SEC. 10. If the expenses of a Section exceed the appropriation made by the Board, the difference may be made up by voluntary contributions from the members of said Section, if it shall so determine. The Institute shall not be responsible for the debts of any Section.

SEC. 11. The Board reserves the right to cancel a Section, or readjust its territory.

SEC. 12. Papers presented at Local Sections, and discussions thereon if reported, shall be the property of the Institute. They shall be submitted to the Publication Committee and published in the *Bulletin* or *Transactions* or both, if approved. Such papers shall not be published elsewhere *in extenso* without permission of the Board. The reading of a paper before a Local Section shall not carry with it the right of publication in the *Bulletin* or *Transactions* of the Institute.

SEC. 13. Neither the author of a paper presented to a Local Section nor the Local Section shall have the right to reprint a paper or publish it in advance of the meeting without obtaining the permission of the Publication Committee of the Institute, which may refuse, or determine the details of such permission. Nothing herein shall forbid the abstracting of a paper by the press after its presentation before a Local Section.

SEC. 14. The Institute shall print advance copies of papers offered to Local Sections, in order to facilitate discussion thereon, provided that such papers are approved for such advance publication by the Chairman or Secretary of the Local Section and by the Publication Committee of the Institute.

SEC. 15. Papers read before a Local Section may also be offered for reading or discussion at general meetings of the Institute, and shall be given equal standing with the other papers on the program of said meeting, when approved by the Publication Committee.

SEC. 16. Each Local Section shall transmit promptly to the Secretary of the Institute announcements of its proposed meetings and an abstract of its proceedings, including the names of authors and titles of all papers read before it, for the purpose of preparing a report thereon to be published in the *Bulletin* of the Institute, and for the purpose of enabling the Board of Directors to comply with sections 17 and 18 of these regulations.

SEC. 17. The By-Laws and regulations of Local Sections shall be subject to the approval of the Board of Directors.

SEC. 18. No action shall be taken by a Section which shall contravene the Constitution and By-Laws of this Institute.

XIX. TECHNICAL COMMITTEES.

SEC. 1. A Committee on Iron and Steel, in charge of the interests of the Institute in this branch of its activities, is hereby established.

SEC. 2. Other Technical Committees may be authorized by the Board of Direc-

tors, charged with the consideration of such topic or topics as the Board may assign, and having the duty of providing papers and discussions upon such topics for the meetings of the Institute.

SEC. 3. The Board of Directors shall designate the officers thereof, who shall hold office for one year or until their successors are appointed.

SEC. 4. The officers of a Technical Committee shall be a Chairman, a Vice-Chairman, and a Secretary.

SEC. 5. The Committee may adopt such rules as are necessary for the convenient conduct of its affairs, subject to the approval of the Board of Directors, or its Executive Committee.

XX. AFFILIATION OR CONSOLIDATION WITH OTHER SOCIETIES.

The Board of Directors may arrange conditions for the affiliation or consolidation with the Institute of any regularly organized group of engineers or any engineering society, whether heretofore or hereafter incorporated or otherwise, which by Constitution, By-Laws, and practice is in accord with the aims of this Institute.

XXI. AFFILIATED STUDENT SOCIETIES.

SEC. 1. Any society of undergraduates at a technical school, comprising students in any branch of engineering, metallurgy, chemistry, geology, etc., may be recognized by the Board, in its discretion, as an Affiliated Student Society, and its name, together with the names of its President and its Secretary, may be published in the *Bulletin* or the Year Book of the Institute. The individual members of such a society will not be catalogued as individual Members of the Institute, but may at any time be proposed for election as Junior Members of the Institute, in the usual way, and receive the *Bulletin* of the Institute, when elected, upon payment of five dollars annual dues.

SEC. 2. Technical papers presented before any such society, and recommended by the officers thereof, will be received and considered by the Publication Committee as if offered by Members of the Institute, and, if accepted, will be published, with the names of their authors, and with due acknowledgment to the said Affiliated Student Society. The members of such societies shall be welcome to use the facilities of the office and library of the Institute.

SEC. 3. Upon the request of the Secretary or other designated officer of any such society, accompanied with the necessary remittance, not more than three copies of the *Bulletin* for one year shall be sent postpaid to him at the reduced price given to public libraries. One volume of *Transactions* may be similarly furnished, at the price charged to Members of the Institute for extra copies.

SEC. 4. The conditions and details of this relation between the Institute and Affiliated Student Societies may be changed at any time in the discretion of the Board of Directors of the Institute; and the relation itself may be terminated at any time by the action of either party.

XXII. AMENDMENTS.

SEC. 1. Amendments to these By-Laws may be made by vote of a majority of the Directors, provided the amendments have been proposed by a member of the Board at a previous meeting of the Board and copies of the same sent to all members of the Board at least 20 days before the meeting at which they are to be voted on, and an expression of opinion invited thereon. At a later meeting they may be amended in form, but not in substance.

Sec. 2. Any Member or Associate of the Institute in good standing may propose amendments or additions to these By-Laws by sending such in writing to the Secretary. The proposal shall be brought before the Board at its next meeting, and, if sponsored by any one member of the Board, shall pass through the course required in the previous section. If the proposal is not so sponsored, it shall be returned to the proposer.

Sec. 3. Amendments or additions to these By-Laws may be proposed by sending the same in writing to the Board of Directors so that they are received by them at least sixty days before a business meeting of the Institute, and providing they be endorsed by the signatures of at least twenty Members or Associates in good standing at the time.

It shall be the duty of the Board of Directors to have such proposals printed in the next succeeding issue of the *Bulletin*, and to provide proper facilities for a vote of the membership thereon by letter ballot before the said business meeting. If favored by a majority of the ballots cast, the amendments shall be reported at the said business meeting as adopted, and shall thereupon become effective.

ANNUAL BUSINESS MEETING.

At the Annual Business Meeting of the Institute, held Feb. 20, 1912, the following officers were elected:

COUNCIL.

President of the Council.

JAMES F. KEMP,	New York, N. Y.

(Term expires February, 1913.)

Vice-Presidents of the Council.

KARL EILERS,	New York, N. Y.
W. LINDGREN,	Washington, D. C.
BENJAMIN B. THAYER,	New York, N. Y.

(Term expires February, 1914.)

Secretary of the Council.

JOSEPH STRUTHERS,	New York, N. Y.

(Term expires February, 1913.)

Councilors.

JOHN H. JANEWAY, JR.,	New York, N. Y.
SIDNEY J. JENNINGS,	New York, N. Y.
JOSEPH W. RICHARDS,	So. Bethlehem, Pa.

(Term expires February, 1915.)

DIRECTORS.

E. B. KIRBY,	St. Louis, Mo.
CHARLES F. RAND,	New York, N. Y.
GEORGE C. STONE,	New York, N. Y.

(Term expires February, 1915.)

It was voted that action upon the proposed amendments to the Constitution, presented by a special Committee, Messrs. Joseph W. Richards, E. Gybbon Spilsbury, and Theodore Dwight, approved by the Board of Directors and the Council, and circulated among the members, be deferred to an adjourned session of this meeting, to be held June 3, 1912, or on such date thereafter as may be mutually agreed upon between the Committee and the Directors of the Institute. (These proposed amendments are on file in the Archives of the Institute).

Voted that Messrs. James F. Kemp, C. R. Corning, George C. Stone, W. H. Nichols, Jr., and A. R. Ledoux, be appointed a Committee of Five, with power to fill vacancies among their

number, to investigate all of the affairs and operations of the Institute, to see whether an increase in dues cannot be avoided, and to report to the Board of Directors by May 1, 1912, making such suggestions as they may deem best for the welfare of the Institute, and that said Board of Directors shall cause such report to be printed and distributed at once by mail to each member, so that it may be considered and acted upon at an adjournment of this meeting. Also, that this Committee be directed and empowered to engage, at the expense of the Institute, independent clerical and auditing assistants.

The meeting adjourned to June 3, 1912. The Board of Directors and the Committee of Five, acting jointly, postponed the adjournment to Oct. 7, 1912.

Session, Oct. 7, 1912.—The Secretary reported that 53 members were present in person, 1,021 by proxy to Messrs. Hunt, Eilers, and Kirchhoff, and 275 by proxy to Messrs. Corning and Finlay. Total, 1,349.

Prof. Joseph W. Richards, *Chairman* of the Special Committee of the Board and Council, which presented certain proposed amendments to the Constitution on Feb. 20, 1912, action on which had been postponed to the Adjourned Business Meeting, withdrew these proposed amendments.

Certain proposed amendments to the Constitution were presented by Messrs. C. R. Corning and George C. Stone, to be voted upon at the Annual Business Meeting of the Institute, Feb. 18, 1913.*

Prof. Joseph W. Richards presented proposed amendments to two articles of the Constitution, to be voted upon at the Annual Business Meeting of the Institute, Feb. 18, 1913.*

Prof. Joseph W. Richards, in behalf of the Special Committee of the Directors and Council appointed to confer with the Committee of Five, advised that conferences had been held, both by these Committees, and by the Directors and Council and the Committee of Five, at a special meeting, Oct. 4, 1912, and that while agreement had been reached in some instances, the majority of the Directors and Council did not approve of the articles proposed by Messrs. Corning and Stone.

It was moved that these and any other amendments to the

* On file in the Archives of the Institute.

Constitution which may be proposed to-day be printed and sent to the membership at large for information, and it was so ordered.

Certain proposed amendments to the By-Laws * were presented by Messrs. C. R. Corning and George C. Stone. After general discussion it was

Voted: To print and send to the membership at large these proposed amendments to the By-Laws; also a brief circular letter summarizing the chief points at issue.

Voted: That it is the sense of this meeting that the dual system of government should be abolished; and to that end the Committee on Amendments to the Constitution (Messrs. Richards, Kirchhoff, and Rand) be requested to take action; and that the Council be directed to take action towards unifying the two governing bodies of the Institute; or else towards defining with absolute clearness the restrictive functions of each body.

Mr. C. F. Rand presented the following resolutions:

Resolved, That the report of the Committee of Five, which has already been received and distributed to members, be now filed with the accompanying documents in the archives of the Institute, and further

Resolved, That the thanks of the Institute are due and are hereby extended to C. R. Corning, *Chairman;* J. F. Kemp, G. C. Stone, W. H. Nichols, Jr., and A. R. Ledoux, the members of the Committee of Five, for their services in the preparation of their report.

Resolved, That the Committee be, and the same is, hereby discharged.

These resolutions were seconded by C. Kirchhoff as one of the representatives of more than one thousand members of the Institute; and the vote in favor was unanimous.

The meeting was adjourned to Nov. 12, 1912.

Session, Nov. 12, 1912.—Prof. Joseph W. Richards, in behalf of the Special Committee of the Board of Directors and the Council (Messrs. Richards, Kirchhoff, and Rand), withdrew the two proposed amendments to the Constitution offered by the said Committee at the Adjourned Annual Business Meeting of Oct. 7, 1912.

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* On file in the Archives of the Institute.

Professor Richards presented a resolution, seconded by Mr. Kirchhoff, making a change in the present Certificate of Incorporation, so that it would conform to the proposed changes in the Constitution and By-Laws; said resolution was unanimously adopted.

Professor Richards presented another amendment to the Constitution, which had been purposely omitted from the printed list, but had been requested by many members, namely, the creation of a "Class of Fellows." This question was freely discussed at the Cleveland meeting, and in view of the diversity of opinion it was decided to present it in separate form for separate action of the membership at large at the Annual Business Meeting, Feb. 18, 1913.

A special vote of thanks was passed to Dr. Albert R. Ledoux for the valuable assistance rendered by him to the Special Committee in the preparation of the proposed amendments to the Constitution and By-Laws.

PROCEEDINGS OF THE BOARD OF DIRECTORS FOR THE YEAR 1911.

The following acts of the Board of Directors of the Institute during the past year are here published for the information of members and associates:

Meeting, Feb. 21, 1911.—The following officers were elected to serve during the ensuing year:

JAMES GAYLEY, *President.*
JAMES DOUGLAS, *Vice-President.*
R. W. RAYMOND, *Secretary.*
FRANK LYMAN, *Treasurer.*

The following Standing Committees were appointed:

Finance: Charles Kirchhoff, Theodore Dwight, Albert R. Ledoux.

Library: R. W. Raymond, James F. Kemp, Charles H. Snow.

The resignation of James Douglas, as representative of the Institute on the John Fritz Medal Board of Award (term expires 1914), was received, accepted, and ordered transmitted to the Secretary of the Board of Award. Mr. R. V. Norris was appointed to fill the vacancy when created.

Mr. Charles Kirchhoff, whose term on the John Fritz Medal Board of Award had just expired, was re-elected to succeed himself (term expires 1915).

The sum of $2,500, or as much thereof as may be needed, was appropriated for library purposes (exclusive of salaries) for the year 1911.

Meeting, Mar. 31, 1911.—The resignation of Dr. Rossiter W. Raymond as Secretary of the Board, member of the Board, and member of the Library Committee, was accepted and Dr. Joseph Struthers was elected to fill the vacancies thus created.

The appointment by the Council of Dr. Raymond as Secretary Emeritus of the Council was confirmed.

In joint action with the Council the following Committee of Fifteen was appointed to prepare an official announcement of the resignation of Dr. Raymond, and the resultant change in the administrative force of the Institute, said announcement to be published in the *Bulletin* for the information of members:

Robert H. Richards, Past President of Council.
E. G. Spilsbury, Past President of Council.
Charles Kirchhoff, { Past President of Council. / President of Council, 1911.
James Douglas, Past President of Council.
E. E. Olcott, Past President of Council.
Albert R. Ledoux, Past President of Council.
James Gayley, { Past President of Council. / President of Board of Directors.
Robert W. Hunt, Past President of Council.
John Hays Hammond, Past President of Council.
D. W. Brunton, Past President of Council.
Anton Eilers, Past Vice-President of Council.
W. L. Saunders, Past Vice-President of Council.
B. B. Lawrence, Vice-President of Council.
J. W. Richards, Vice-President of Council.
Albert Sauveur, Vice-President of Council.
Frank Lyman, Treasurer, Board of Directors.

Meeting, April 28, 1911.—The report of the Special Committee on Increase of Dues of the Institute, appointed at the Annual Business Meeting of the Institute, Feb. 21, 1911 (Messrs. Joseph W. Richards, E. G. Spilsbury, and Theodore Dwight), was accepted.

The following Special Committee to prepare a proper form of proxy to accompany the recommendation to increase the dues of the Institute was appointed: Messrs. Joseph W. Richards, E. G. Spilsbury, and Theodore Dwight.

The following Special Committee of Nine, to consider and report on the change of name of the Institute to American Institute of Mining and Metallurgy, was appointed: Messrs. James Gayley, A. R. Ledoux, and James F. Kemp, of the Board; Messrs. C. Kirchhoff, B. B. Lawrence, and Gardner F. Williams, of the Council; Messrs. R. W. Raymond, W. R. Ingalls, and W. L. Saunders, of the membership at large.

A check for $10,913.30, sent by James Douglas as a contribution to the Land Fund from Phelps, Dodge & Co., was received, and an appropriate letter of thanks was ordered to be sent to Phelps, Dodge & Co. for their munificent gift.

The resignation of Prof. Charles H. Snow as member of the

Board, and of the Library Committee, was accepted, and Prof. Arthur L. Walker was elected to fill the vacancies thus created.

Meeting, May 19, 1911.—The report of the Special Committee of Nine recommending the change of name of the Institute to the American Institute of Mining and Metallurgy was approved, and ordered to be printed and distributed to the membership.

On the unanimous recommendation of the Council, Prof. R. H. Richards and Dr. R. W. Raymond were unanimously elected Honorary Members of the Institute.

The Assistant Treasurer presented a statement of receipts and expenditures of the Institute for the current year to May 1, which was accepted and placed on file.

Meeting, June 16, 1911.—The report of the Special Committee (Messrs. J. W. Richards, E. G. Spilsbury, and Theodore Dwight), appointed Apr. 28, to prepare announcement concerning increase of dues, was accepted.

The Assistant Treasurer presented a memorandum of receipts and expenditures of the Institute for the month of May, which was accepted and placed on file.

Meeting, Aug. 29, 1911 (*Special*).—The reconvening of the adjourned Annual Business Meeting of the Institute was authorized to be called for Sept. 20, 1911.

Meeting, Sept. 20, 1911 (*Adjourned Annual Business Meeting*).—In conformity with the provisions of Article XII. of the Constitution, written notice was formally presented that certain proposed amendments to the Constitution would be offered for vote at the Annual Meeting, Feb. 20, 1912, as was also an additional amendment concerning a Nominating Committee * presented in person by Mr. E. G. Spilsbury.

Meeting, Sept. 20, 1911.—The proposed amendments to the Constitution, as published in General Announcement No. 3, Sept. 1, 1911, and the proposed amendment by E. G. Spilsbury, individually presented at the adjourned Annual Business Meeting, Sept. 20, 1911, were approved.

The Assistant Treasurer presented a memorandum of receipts and expenditures of the Institute for June, July, and August, which was accepted and placed on file.

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* On file in the Archives of the Institute.

Meeting, Oct. 20, 1911.—The matter of the proper disposal of the large stock of back volumes of the *Transactions*, and of separate pamphlets of papers, was referred to the Library Committee, with the request to report at a later meeting of the Board.

It was voted to anticipate the adoption of the proposed rule to create a Nominating Committee.

Meeting, Nov. 17, 1911.—The following appointments to the Nominating Committee were reported: Messrs. E. G. Spilsbury, R. V. Norris (later A. S. Dwight), J. R. Finlay, L. D. Huntoon, and C. P. Perin.

The form of circular prepared by the Nominating Committee for distribution to the membership was approved.

The report of the Library Committee recommending special prices for various sets of the *Transactions* * was approved.

The President of the Council reported that the Post Office authorities had placed the *Bulletin* of the Institute on the Second Class list, which will insure a saving in postage of approximately $1,500 per year. In connection with the action of the Post Office authorities, a special vote of thanks was passed for the friendly co-operation and active interest of Mr. Charles L. Parsons, Secretary of the American Chemical Society.

Meeting, Dec. 15, 1911.—Messrs. E. E. Olcott and George W. Maynard were appointed proxies for the vote on the proposed amendments to be acted upon at the Annual Business Meeting, Feb. 20, 1912.

The Assistant Treasurer presented a memorandum of receipts and expenditures of the Institute to Dec. 1, which was accepted and placed on file.

Meeting, Jan. 12, 1912.—The report of the Nominating Committee was accepted and ordered to be sent to every member and associate entitled to vote. Said nominations were:

For President,	JAMES F. KEMP,
For Vice-Presidents,	BENJAMIN B. THAYER,
	KARL EILERS,
	WALDEMAR LINDGREN.
For Councilors,	JOSEPH W. RICHARDS,
	JOHN H. JANEWAY, JR.,
	SIDNEY J. JENNINGS.
For Secretary,	JOSEPH STRUTHERS.

* On file in the Archives of the Institute. (See pp. liv and lv of this volume.)

The form of the General Announcement (No. 1 of 1912) concerning the proposed amendments to the Constitution, the form of the proposed amendments, and the form of proxy for vote on the proposed amendments, were accepted and authorized for publication.

The form of proxy for the election of officers, etc., at the Annual Meeting, Feb. 20, 1912, was accepted, and Messrs. Leonard Waldo and Bradley Stoughton were appointed to act as proxies for said election, and for such other business as may come before the Annual Meeting, Feb. 20, 1912, excepting the action on the proposed amendments.

The report of the Treasurer giving an account of the receipts and disbursements for the year 1911, which had been audited by Barrow, Wade, Guthrie & Co., C. P. A., was adopted for presentation at the Annual Business Meeting of the Institute, Feb. 20, 1912.

The Treasurer was authorized to charge off from the Furniture and Fixtures account 10 per cent. per annum for depreciation.

The schedule of salaries for the calendar year 1912 for employees (not including the Library service, which is administered by the United Engineering Society), presented by the Assistant Treasurer, was accepted subject to ratification by the Board of Directors as constituted at the meeting directly following the Annual Business Meeting of the Institute, Feb. 20, 1912.

An appropriation of $1,500 (or as much thereof as may be required) was authorized for Library purposes (not including salaries) for the year 1912.

Prof. James F. Kemp was elected a representative of the Institute on the Board of Trustees of the United Engineering Society to succeed Mr. E. E. Olcott, who retired January, 1912, by limitation.

The report of the Land Fund Committee was presented by Theodore Dwight, Treasurer, showing receipts during the year of $11,273.05, and disbursements of $11,000 on account of principal and mortgage, leaving a balance of $273.05. The original obligation was $288,000, of which the Institute has paid $114,000 of the principal, besides meeting the current interest of 4 per cent. upon the unpaid amount. Outstanding subscriptions to the amount of $6,000 and the cash on hand will reduce the debt to $67,726.95.

Messrs. Charles Kirchhoff and James F. Kemp were appointed a Committee to consider the advisability of increasing the office force and report recommendations at the next meeting of the Board.

Meeting, Feb. 9, 1912.—The circular * signed by Messrs. Stone, Corning, Ingalls, and others, under date of Feb. 3; also a letter * from Waldemar Lindgren, of Washington, concerning the proposed amendments to the Constitution, were presented, and after full deliberation, it was the sense of the meeting to invite the signers of the circular and Mr. Lindgren to appoint a Committee with power to call for any information or documents in the possession of the officers of the Institute at as early a date as possible (and in any event, prior to the Annual Business Meeting).

The Assistant Treasurer presented a report of the receipts and disbursements of the Institute for the month of January, 1912, and the corresponding month of 1911, which was accepted and ordered on file.

The Secretary submitted a brief abstract of the acts of the Board of Directors during the past year, together with a report of the Library Committee, and a brief covering the general meetings of the Institute, publications, and membership. The publication of these reports, for the information of the members, was authorized.

JOSEPH STRUTHERS,
Secretary.

Treasurer's Report for the Calendar Year 1911.

The following statement of receipts and expenditures from Jan. 1 to Dec. 31, 1911, was authorized to be published for the information of members and associates by the Board of Directors Jan. 12, 1912.

RECEIPTS.

Balance from statement of January, 1911,		$3,938.17
Annual dues,	$35,619.81	
Life memberships,	1,930.00	
Initiation fees,	1,990.26	
Binding of *Transactions*,	3,271.48	
Sale of publications, electrotypes, advertising, and miscellaneous receipts,	17,368.12	
Interest on bank deposits,	168.23	
		64,286.07

* On file in the Archives of the Institute.

DISBURSEMENTS.

Printing Vol. XLI. of the *Transactions*, *Bulletin*, extra pamphlets, and advertising expenses, etc.,	$15,966.73	
Printing circulars and ballots,	745.18	
Binding Vol. XLI. of the *Transactions*,	3,495.00	
Binding miscellaneous volumes,	137.15	
Engraving and electrotyping,	1,023.31	
Secretary's department, including clerks, stenographers, and expenses of editing and proof-reading, and special assistance in connection with meetings,	10,333.91	
Treasurer's department, including collection of dues, shipping, etc.,	6,881.25	
Library,	2,356.21	
Postage,	4,470.91	
Stationery,	483.09	
Express and freight,	1,139.06	
Telephone,	271.90	
Telegrams, cables, carfares, etc.,	121.94	
Office supplies, repairs,	415.53	
Refunding miscellaneous payments,	56.63	
Insurance premiums (fire and surety),	369.12	
Collection charges,	40.51	
Extra clerical assistance,	195.67	
Special stenographers and expenses of meetings,	1,914.38	
Auditing,	115.00	
Sundry expenses,	37.00	
		$50,569.48
Interest at 4 per cent., for 1911, on unpaid balance of land mortgage on 25 to 33 West 39th St. ($85,000, January 1, 1911, reduced to $74,000 April 27, 1911),	3,133.00	
Quota of current expenses of building 25 to 33 West 39th St.,	4,500.00	
		7,633.00
Special editing, new volume of Genesis of Ore-Deposits,	150.00	
Special editing, printing and binding of Index to Vols. XXXVI. to XL.,	775.80	
		925.80
Library additions of books, periodicals, etc., binding of exchanges, and stationery (expenditure from appropriation of $2,500),		811.83
Furniture and fixtures,		114.96
Balance,		4,231.00
		$64,286.07

(Signed) FRANK LYMAN, *Treasurer*.

NEW YORK, N. Y., January 20, 1912.

We have examined the above statement, compared it with the books and vouchers and find same correct.

(Signed) BARROW, WADE, GUTHRIE & Co.,
Certified Public Accountants.

SPECIAL FINANCIAL REPORT CONCERNING THE ENGINEERING SOCIETIES BUILDING.

Many of the members of the American Institute of Mining Engineers, not resident in the vicinity of New York, in spite of the regular annual reports and the detailed statements which were sent out when the gift of the Engineering Societies Building was received from Mr. Andrew Carnegie, are unfamiliar with the relations between our Institute and the United Engineering Society, in whose name the title stands. It has therefore been deemed best by the Council and Directors to submit the following statement:

The title is held by the United Engineering Society as Trustees for our Institute, the Electrical, and the Mechanical Engineers, each owning one-third. Each Society appoints representatives on this Board of Trustees, and is responsible for one-third of the cost of the land upon which the building is erected and up to one-third of the running expenses. So much of the building as is not needed for the purposes of the Societies is allotted to other technical or allied organizations, each contributing proportionately to the running expenses. All three Societies have the privilege of admitting to such part of their own floors as they do not for the present require, other societies or institutions not of a business character.

The cost of the building was $1,050,000, given by Mr. Carnegie. The land cost the three Founder Societies $540,000, and has greatly appreciated in value. It is understood that towards the cost of the land, the Electrical Engineers have received gifts covering their entire proportion of the land fund. The Mechanical Engineers, by gifts and by the issuing of certificates of indebtedness, have raised their entire proportion. Generous members of our Institute have subscribed $111,700 in gifts, leaving but $68,000 to be raised by us. There is a mortgage upon the land, bearing interest at 4 per cent. It is necessary that our Institute, if it does not receive further cash gifts, shall arrange to pay off its proportion before the expiration of the mortgage. In the last resort, interest-bearing debentures might be issued, to be paid off by annual sums to be obtained from a surplus in the Institute's regular receipts.

If extinguished in ten years, this, with interest at 4 per cent., would call for about $75,000.

It is for this reason, among others, that the Directors and Council approved of the suggestion to increase the dues. The running expenses alone on the present basis could be cared for by the present receipts from dues, etc. The Institute at present is required to pay to the Trustees of the Engineering Society about $4,500 annually towards the cost of maintenance, which is, for office space, very low for the locality and for the space required. The United Engineering Society has to date accumulated from its revenues a fund for depreciation and reserve of $25,000, so that the annual assessment of $4,500 for each Founder Society may be considered the probable maximum. In addition to its regular receipts, the Institute has received from participants in the occupancy of its floor the following sums:

1909	1910	1911
$660	$2,809.33	$1,490.45

The Institute is this year deriving on the present basis an income of $750 from this source.

The net receipts and expenditures of the Institute for 1906 to 1911, inclusive, have been as follows:

	Net Receipts.	Expenditures.
1906,	$51,680.39	$41,127.24 (*a*)
1907,	55,549.42	55,181.45 (*b*)
1908,	55,293.27	48,311.44 (*b*)
1909,	53.750.91	50,023.00 (*c*)
1910,	51,485.39	49,575.81 (*d*)
1911,	57,282.18	53,856.35 (*e*)
	$325,041.56	$298,075.29

NOTE.—

(*a*) Including $2,500 rent at 99 John Street.
(*b*) Including $4,500 assessment considered as equivalent to rent.
(*c*) Including $4,500 assessment less $660 released.
(*d*) Including $4,500 assessment less $2,809.33 released.
(*e*) Including $4,500 assessment less $1,490.45 released.

In 1907 there was an extraordinary expenditure of $2,625.90 for the large index of our thirty-five volumes of *Transactions* and $3,000 additional for equipment of the new quarters and moving.

The net receipts, 1906 to 1911, were,	$325,041.56
The ordinary expenditures, 1906 to 1911, were,	298,075.29
Surplus,	$26,966.27

Under the conditions of revenue and outlay, covering a series of six years, the Institute finances showed a book surplus of $26,966.27.

The contributions from the Institute to the Engineering Building during this period of 1906 to 1911, inclusive, have been as follows:

Interest payments,		$24,062.99
Total assessments,	$31,700.00	
Less charged for office space,	22,500.00	
Net assessments,		9,200.00
Payment on principal,		15,000.00
		$48,262.99

These drafts upon the Institute have been met in the following manner:

Sale of securities,		$18,708.76
Surplus,		26,966.27
Balance in bank reduced		
From January 1, 1906,	$6,818.96	
To January 1, 1912,	4,231.00	
		2,587.96
		$48,262.99

The total obligations assumed on account of the Engineering Societies Building on the part of the Institute to the end of 1911 have been as follows:

One-third original mortgage,	$180,000.00
Advance payment under Founders' agreement,	8,000.00
1906 interest paid from Land Fund,	3,800.00
1907 interest paid from Land Fund,	2,600.00
Interest payments by Institute,	24,062.99
Net assessments,	9,200.00
	$227,662.99

Besides the gifts of $105,700 made to us outright, there is an additional $6,000 which has been pledged, and upon which the donors are paying us interest until they see their way to liquidate the principal.

PROCEEDINGS OF THE COUNCIL FOR THE YEAR 1911.

The following acts of the Council of the Institute are here published for the information of members and associates:

Meeting, Feb. 21, 1911.—Messrs. W. L. Saunders and George C. Stone were appointed delegates to represent the Institute at the Eighth International Congress of Applied Chemistry in connection with the sessions to be held in New York in September, 1912.

Meeting, Mar. 31, 1911.—The resignation of Dr. Rossiter W. Raymond as Secretary of the Council and as a member of the Committee on Membership was accepted, and Dr. Joseph Struthers was appointed to fill the vacancies thus created for the unexpired term.

Dr. Raymond was appointed Secretary Emeritus of the Council of the Institute.

In joint action with the Directors the following Committee of Fifteen was appointed to prepare an official announcement of the resignation of Dr. Raymond and the resultant change in the administrative force of the Institute, said announcement to be published in the *Bulletin* for the information of members:

Robert H. Richards, Past President of Council.
E. G. Spilsbury, Past President of Council.
Charles Kirchhoff, { Past President of Council. / President of Council, 1911.
James Douglas, Past President of Council.
E. E. Olcott, Past President of Council.
Albert R. Ledoux, Past President of Council.
James Gayley, { Past President of Council. / President of Board of Directors.
Robert W. Hunt, Past President of Council.
John Hays Hammond, Past President of Council.
D. W. Brunton, Past President of Council.
Anton Eilers, Past Vice-President of Council.
W. L. Saunders, Past Vice-President of Council.
B. B. Lawrence, Vice-President of Council.
J. W. Richards, Vice-President of Council.
Albert Sauveur, Vice-President of Council.
Frank Lyman, Treasurer of Board of Directors.

Meeting, April 28, 1911.—The following Special Committee was appointed to consider the question of establishing Local Sections of the Institute: Charles F. Rand, E. G. Spilsbury, A. L. Walker, Joseph W. Richards, and Karl Eilers.

The following Special Committee was appointed to consider the establishment of a Committee on Publications: Joseph W. Richards, A. Sauveur, and S. B. Christy.

Meeting, May 19, 1911.—Prof. Robert H. Richards and Dr. Rossiter W. Raymond were unanimously recommended to the Board of Directors for election to Honorary Membership.

Prof. John D. Irving was appointed a representative of the Institute to the American Association for the Advancement of Science, to fill the vacancy caused by the death of Dr. S. F. Emmons.

The report of the Committee on Local Sections was approved and ordered to be printed, and sent to the membership of the Institute.

Permission was granted for the establishment of Local Sections of the Institute at San Francisco and at New York, under the approved regulations.

Meeting, June 16, 1911.—A Committee of Five (Messrs. Charles Kirchhoff, Joseph Struthers, James F. Kemp, Charles F. Rand, and Frank Lyman), was appointed to confer with a similar Committee from the Mining and Metallurgical Society of America, with a view to ascertaining the possibility of consolidation of the two societies.

The Regulations for the Committee on Publications,* as presented by the Special Committee, Joseph W. Richards, *Chairman*, were accepted.

Meeting, Sept. 20, 1911.—The report of the Special Committee of Five appointed to confer with a similar Committee from the Mining and Metallurgical Society of America, stated that two joint conferences had been held, and after a detailed discussion of the views and work of the A. I. M. E., and the M. & M. S. of A., and the position taken by the latter that its identity and organization must be continued, it was resolved that for the present the proposed affiliation of the Institute and the Society be carried out to the extent of having joint meet-

* On file in the Archives of the Institute.

ings, both of Local Sections and of the parent bodies. This report was approved and accepted.

Permission was granted for the establishment of a Local Section in Boston under the approved regulations.

The following Committee on Publications was appointed:

The Secretary-Editor of the Institute, *Chairman;* David W. Brunton, Samuel B. Christy, Albert L. Colby, Nathaniel H. Emmons, Charles H. Fulton, James Gayley, H. O. Hofman, Henry M. Howe, Walter R. Ingalls, James F. Kemp, R. V. Norris, Edward D. Peters, Rossiter W. Raymond, Joseph W. Richards, Robert H. Richards, Albert Sauveur, Henry L. Smyth, and Arthur L. Walker.

The following delegates or representatives were appointed: William J. Chalmers and Robert N. Dickman, to the American Mining Congress, Chicago, Oct. 24–28, 1911.

H. V. Winchell, Installation of George E. Vincent as President of the University of Minnesota, Minneapolis, Oct. 18, 1911.

J. W. Malcolmson, E. E. Howard, P. E. McMullen, and P. N. Moore, to the Third National Conservation Congress, Kansas City, Sept. 25–27, 1911.

Capt. Robert W. Hunt (twice Past President of the Council) was appointed the official representative of the Institute for the San Francisco meeting, and the excursion to Japan.

Meeting, Oct. 20, 1911.—The application for the establishment of a Local Section of the Institute in Spokane, Wash., was granted under the approved regulations.

Meeting, Dec. 15, 1911.—Applications for enrollment as Affiliated Student Societies were granted to the Scientific Society of the Colorado College of Mines, and the Pick and Shovel Club of the Case School of Applied Science.

Robert W. Forsyth was appointed to represent the Institute at the meeting of the International Association for Testing Materials, September, 1912.

Prof. James F. Kemp was elected a member of the John Fritz Medal Board of Award for the period Jan. 1, 1912, to Jan. 1, 1916.

Messrs. Charles Kirchhoff and Joseph Struthers, as President and Secretary of the Council, were appointed representatives of the Institute on the Joint Conference Committee of the Four National Engineering Societies.

Meeting, Jan. 12, 1912.—The By-Laws of the Spokane Local Section of the Institute were approved.

Mr. Charles Kirchhoff was elected to represent the Institute at the 1912 Congress on Accident Prevention of the Congrès Technique International, at Milan, Italy.

Messrs. Samuel B. Christy, William C. Ralston, and Edwin T. Blake were appointed delegates to represent the Institute at the International Engineering Congress, San Francisco, 1915.

The following honorary delegates were appointed to represent the Institute at the annual meeting of the Canadian Mining Institute, Toronto, March 6, 7, and 8, 1912: Messrs. John Birkinbine, H. M. Chance, C. R. Corning, James Douglas, Theodore Dwight, W. E. C. Eustis, J. R. Finlay, F. Lynwood Garrison, Walter R. Ingalls, William Kelly, James F. Kemp, Benjamin B. Lawrence, Albert R. Ledoux, Ambrose Monell, Henry S. Munroe, R. V. Norris, Edward W. Parker, Robert H. Richards, E. Gybbon Spilsbury, H. H. Stoek, Joseph Struthers.

INSTITUTE ACTIVITIES FOR THE YEAR 1911.

Meetings of the Institute.

There were held during the year 1911 two meetings of the Institute for the presentation and discussion of technical papers —the 100th meeting, at Wilkes-Barre, Pa., June 6–10, and the 101st meeting, at San Francisco, Cal., October 10–17.

A detailed record of the proceedings of these meetings, including a description of the entertainments and excursions connected therewith, has been published and duly distributed to the membership—the Wilkes-Barre meeting in *Bulletin* No. 55, July, 1911, pages 575 to 594; the San Francisco meeting in *Bulletin* No. 59, November, 1911, pages v. to xxxviii.; and the excursion to Hawaii and Japan in *Bulletin* No. 61, January, 1912, pages 1 to 102.

Local Sections.

During the year, local sections have been formed at New York, Spokane, and Boston, and meetings have been held as recorded in the *Bulletin.*

Affiliated Student Societies.

Ten organizations of undergraduates at technical schools and universities were recognized as Affiliated Student Societies during the year 1911. The total number of Societies enrolled is 21, representing the following institutions: Yale University, Sheffield Scientific School; University of Illinois, University of Nevada, University of Wisconsin, Lehigh University, University of Minnesota, Massachusetts Institute of Technology, University of Kansas, University of Idaho, State College of Washington, University of Texas, Ohio State University, Stanford University, Columbia University, University of California, Tufts College, University of Washington, Iowa State College, Missouri School of Mines, Case School of Applied Science, Colorado School of Mines.

Publications.

Transactions.—Volume XLI. of the *Transactions*, an octavo of 1004 pages, comprising 55 papers and 10 discussions presented during the year 1910, was issued and distributed to the membership in June.

Bulletin.—Twelve numbers of the *Bulletin* (Nos. 49 to 60), containing the technical papers and discussions of the Institute (in "subject to revision" form) and announcements of general interest to the members of the Institute, such as Library accessions and requirements during the year 1911, notices of meetings of the Institute and other societies, lists of proposed members and associates, changes of address, deaths of members, obituary notices, etc., have been published and distributed promptly throughout the year 1911. The number of pages occupied by technical papers and discussions amounts to 1042, to which are to be added 400 pages of announcements, and 360 pages of advertising matter, making a total of 1802 pages of printed matter.

Membership.

Changes in the membership of the Institute have taken place during the calendar year 1911 as follows:

Two Honorary members, 189 members, and 13 associates have been elected; 14 members have been reinstated; and 3 associates have become members; a total of 221. Against this number should be recorded the deaths of 48 members and 3 associates; the resignation of 94 members and 9 associates; and the dropping from the roll, by reason of non-payment of dues, of 93 members and 5 associates—a total of 252, which shows a net loss in membership of 36. The net loss in membership in 1910 was 74. The total membership on Jan. 1, 1912, was 4,174.

Library Committee.

The following is the Report of the Library Committee for the year ending Feb. 20, 1912. R. W. Raymond, *Chairman*, James F. Kemp, and Charles H. Snow were appointed Feb. 21, 1911. Dr. Raymond resigned from this Committee on Mar. 31, 1911, and was succeeded by Dr. Joseph Struthers. Professor Snow resigned Apr. 28, 1911, and was succeeded by Prof. A. L. Walker.

The Library Committee presented to the Board at the meeting of Nov. 17, 1911, the following schedule of reduced prices for various sets of volumes of the *Transactions*, which was duly approved by the Board.

		Per Set.
I.	200 sets of 5 volumes, bound, from No. 36 (1906) to No. 40 (1910)	$20.00
II.	75 sets of 10 volumes, bound, from No. 31 (1902) to No. 40 (1910) Including the Mexican Volume,	$35.00
III.	50 sets of 20 volumes, bound, from No. 21 (1893) to No. 40 (1910)	$50.00
IV.	25 sets of 30 volumes, bound, from No. 11 (1883) to No. 40 (1910)	$60.00
V.	50 sets of 39 volumes, bound, from No. 1 (1873) to No. 40 (1910) with the exception of No. 10 (1882)	$75.00
VI.	50 sets of 9 volumes, bound, from No. 1 (1873) to No. 9 (1881)	$25.00

Committtee on Membership.

The following is the report of the Committee on Membership for the year ending Feb. 20, 1912. Dr. R. W. Raymond, *Chairman*, Benjamin B. Lawrence, Karl Eilers, Charles F. Rand and Edward L. Young were appointed on the Committee on Membership, Feb. 21, 1911. Dr. Raymond resigned from the Committee on Mar. 31, 1911, and Dr. Joseph Struthers was elected to fill the vacancy for the unexpired term.

During the past year, the Committee met monthly, except in July and August, acted upon 224 cases, and made the following recommendations to the Council: for election as member, 194; for election as associate, 20; for change of status from associate to member, 7.

PROCEEDINGS OF THE BOARD OF DIRECTORS FOR THE YEAR 1912.

The following acts of the Board of Directors of the Institute during the past Institute year are here recorded for the information of the members:

Meeting, Feb. 28, 1912.—The following officers were elected to serve during the ensuing year:

JAMES F. KEMP, *President.*
EDMUND B. KIRBY, *Vice-President.*
GEORGE C. STONE, *Secretary.*
FRANK LYMAN, *Treasurer.*

The following officer was appointed to serve during the ensuing year:

JOSEPH STRUTHERS, *Assistant Secretary* and *Assistant Treasurer.*

The following Standing Committees were appointed:

Finance: Charles F. Rand, James Gayley, George C. Stone.

Library: James F. Kemp, Charles Kirchhoff, George C. Stone.

The following Special Committee was continued:

Land Fund: James Douglas, *Chairman;* Theodore Dwight, *Treasurer;* T. F. Cole, Anton Eilers, James Gayley, John Hays Hammond, Charles Kirchhoff, Albert R. Ledoux, Frank Lyman.

Voted: That the salary list accepted at the joint meeting of the Board and Council, Jan. 12, 1912, subject to approval by the Board at the meeting directly following the Annual Business Meeting, be hereby approved for period of six (6) months.

Meeting, Mar. 22, 1912.—The Treasurer presented a report of the receipts and disbursements for the month of February, 1912, and the corresponding month of February, 1911; also the total receipts and disbursements for January and February, 1912, with the corresponding period of January and February, 1911. The report was accepted and ordered on file.

Meeting, April 8, 1912.—At the request of the Assistant Treasurer, and after general discussion, it was voted that all bills shall be approved by a member of the Finance Committee before payment by the Treasurer or the Assistant Treasurer.

Meeting, May 24, 1912.—The Board of Directors, after care-

ful consideration of the tentative budget of the Finance Committee, adopted certain economies in the salary list of the officers of the Institute.

Meeting, June 28, 1912.—The Assistant Treasurer presented a report of the receipts and disbursements for the month of May, 1912, and the corresponding month of May, 1911, also the total receipts and disbursements for January to May, 1912, with the corresponding period of January to May, 1911. The report was approved and ordered on file.

The Library Committee, Prof. Kemp, *Chairman*, presented a printed form of proposed amendments to the Constitution of the United Engineering Society, as approved by the Library Conference Committee, Feb. 14, 1912, and after discussion it was voted that the recommendations as embodied in the printed document be approved and placed on file.

The Land Fund Committee, through Dr. Douglas, *Chairman*, reported informally that he intended to make a personal appeal to the membership at large for contributions to the Land Fund.

The Special Committee of Five, appointed at the Annual Business Meeting of the Institute, Feb. 20, 1912, submitted its report, which was ordered on file. After discussion it was finally voted that this report be printed as a separate circular to be sent to the members as a private communication.

The following Special Committees were appointed:

Proposed Amendments to the Constitution: Messrs. Joseph W. Richards, James Douglas, and Hennen Jennings.

Changes in the By-Laws: Joseph Struthers, Charles Kirchhoff, Albert R. Ledoux, Karl Eilers, and Edward L. Young.

Life Members and Delinquent Members: Charles F. Rand, James Gayley, and George C. Stone.

Meeting, Sept. 27, 1912.—The Assistant Treasurer presented a report of the receipts and disbursements for the month of August, 1912, and the corresponding month of August, 1911, also the total receipts and disbursements for January to August, 1912, with the corresponding period of 1911. The report was approved and ordered on file.

It was voted to grant the request of Mr. L. K. Armstrong, *Secretary*, that the initiation fees of new members of the Spokane Local Section be applied to defray the expenses of that Section until said amount should reach 25 per cent. of the

vised form of proposed Amendments to the Constitution and By-Laws, and in pursuance of the instruction of the Institute given at the Adjourned Annual Business Meeting, Nov. 12, 1912, the Board of Directors adopted and approved the form of the proposed amendment to the Constitution and By-Laws as prepared by the Attorney of the Institute and filed in the Archives of the Institute. The report of this Special Committee, Messrs. Richards, Kirchhoff, and Rand, was duly approved, and a vote of thanks was passed to the Committee for its arduous labors; also a special vote of thanks was placed on record for the valuable assistance and co-operation of Dr. Albert R. Ledoux in connection with the work of this Committee.

Meeting, Dec. 12, 1912.—The President read a letter which he had drawn up to accompany the new proposed amendments to the Constitution and By-Laws for the membership, which was referred to the President with power.

Mr. Stone, in behalf of Messrs. Corning and Stone, who had submitted certain proposed amendments to the Constitution and By-Laws at the Adjourned Annual Business Meeting, Oct. 7, 1912, withdrew all the proposed amendments with the exception of the following:

Constitution: Art. II., Sec. 2, second paragraph of the proposed new Constitution.

Constitution: Art. III., Sec. 1, third paragraph of the proposed new Constitution.

By-Laws: Art. VI. of the proposed new By-Laws.

By-Laws: An additional By-Law regarding the creation of Divisions.

It was voted that the proxies for the vote on the new Constitution and By-Laws at the Annual Business Meeting, Feb. 18, 1913, be made out in the names of James Douglas and Albert R. Ledoux.

The Assistant Secretary reported that a representative of the Institute on the Board of Trustees of the United Engineering Society must be elected to take the place of Joseph Struthers, whose term of office expires January, 1913, and on motion Joseph Struthers was duly elected to succeed himself for the term of three years.

The Assistant Treasurer presented a report of the receipts and disbursements for the month of November, 1912, and the

corresponding month of November, 1911; together with the aggregate receipts and disbursements for the 11 months, Jan. 1 to Nov. 30, 1912, and the corresponding period of 1911. This report was accepted and ordered on file.

It was moved by Dr. Ledoux and seconded by Mr. Rand that the Secretary be instructed to communicate to the representatives of the Institute on the Committee for the International Engineering Congress to be held in San Francisco, 1915, that this Institute will stand its share, not to exceed $5,000, and it was so voted.

It was further voted that for the next three years, beginning with 1912, $1,000 be set aside each year for the purpose of meeting the Institute's share of the expense of the International Engineering Congress in 1915.

In view of the large numbers of papers presented at the meetings of the Institute during 1912, which exceed the capacity of a single volume of convenient size, the publication of these papers in two volumes of *Transactions* was authorized.

The Assistant Treasurer was authorized to charge off 10 per cent. depreciation from the "Furniture and Fixture Account," for the year 1912.

The Special Committee on Delinquent Members, Mr. Rand, *Chairman*, reported that the list of delinquents had been very carefully scrutinized, and recommended that of the total number now in arrears for three years' dues, 77 be dropped unless they paid their dues before the end of the year; 14 be placed on a suspended list, and 4 be held for subsequent action.

A communication was received from Mr. A. H. Eustis, *Secretary-Treasurer*, of the Boston Local Section of the Institute, requesting that a sum of $50 be contributed by the Institute to meet the debts already incurred by the Section, and on motion duly seconded the payment of said amount was authorized.

Meeting, Jan. 23, 1913.—The President announced the appointment of Messrs. W. R. Ingalls and W. H. Nichols, Jr., as proxies for the vote for members of the Council.

The President recommended the appointment of the present Library Committee, Messrs. James F. Kemp, George C. Stone, and Charles Kirchhoff, together with Charles F. Rand, as four representatives of the Institute to serve with the Secretary on the new Library Board of Control, and it was so ordered.

The Assistant Treasurer presented a comparative statement of receipts and disbursements for December, 1911 and 1912; also the annual report for the year 1912, showing a cash balance on hand Jan. 1, 1913, of $7,350 as compared with $4,231 for Jan. 1, 1912. These reports were accepted and ordered on file.

The Finance Committee presented the estimated budget for 1913, revised to Jan. 17, 1913, which was ordered on file.

The Land Fund Committee, through Dr. Ledoux, reported informally that Dr. Douglas had received contributions of practically $40,000, leaving a balance of $28,000, which amount it is hoped will be realized by the time of the Annual Business Meeting in 1913, in response to a circular letter being sent to the membership, signed by five members of the Institute, requesting 23 additional members to subscribe or be responsible for $1,000 each. The Assistant Treasurer reported that the amount on deposit with the Treasurer of the Land Fund Committee on Jan. 1, 1912, was $273.05; that during the year 1912, $235 was received, making a total balance on hand Jan. 1, 1913, of $508.05.

The report of the Special Committee of Five, Prof. Kemp, *Chairman*, on the proposed affiliation with the Mining and Metallurgical Society of America, was presented and was approved by a majority of the members of the Board present. It was the informal consensus of the opinion of those present that the question of affiliation with the Mining and Metallurgical Society of America should be referred to the membership at large for a vote on the final action.

GEORGE C. STONE,
Secretary.

Treasurer's Report for the Calendar Year 1912.

The following statement of receipts and disbursements from Jan. 1 to Dec. 31, 1912, properly audited by Barrow, Wade, Guthrie & Co., Chartered Accountants, was authorized to be published for the information of members and associates by the Board of Directors, Feb. 14, 1913.

RECEIPTS.

Balance from statement of January, 1912,		$2,231.00
Annual dues,	$37,498.66	
Life Memberships,	2,370.00	
Initiation fees,	2,930.98	
Binding of *Transactions*,	3,583.17	
Sale of publications, electrotypes, advertising and miscellaneous receipts,	15,462.60	
Interest on bank deposits,	224.92	
		62,070.33
		$66,301.33

DISBURSEMENTS.

Printing Vol. XLII. of *Transactions*, *Bulletin*, extra pamphlets, advertising expenses, etc,	$17,740.01	
Printing circulars and ballots,	843.62	
Binding Vol. XLII. of *Transactions*,	3,392.89	
Binding miscellaneous volumes,	597.90	
Engraving and electrotyping,	1,053.76	
Secretary's department, including clerks, stenographers, and expenses of editing and proof-reading,	10,183.33	
Treasurer's department, including collection of dues, shipping, etc.,	5,297.34	
Library,	2,994.10	
Postage,	3,026.52	
Stationery,	641.44	
Express and freight,	1,200.00	
Telephone,	289.80	
Telegrams, cables, carfares, etc.,	105.38	
Office supplies, repairs,	424.15	
Refunding miscellaneous payments,	20.91	
Insurance premiums (fire and surety),	179.00	
Collection charges,	54.79	
Extra clerical assistance,	552.92	
Special stenographers and expenses of meetings:		
San Francisco (1911),	196.16	
N. Y. Annual and adjournments,	516.37	
Cleveland,	324.18	
Int. Asso. for Testing Materials, John Fritz Metal Board of Award, Iron and Steel Division,	296.75	
Auditing,	390.00	
Sundry Expenses,	806.53	
		51,127.85
Interest at 4 per cent. for 1912, on $74,000 unpaid balance of land mortgage on 25 to 33 West 39th Street,	$2,960.00	
Quota of current expenses of building 25 to 33 West 39th Street,	3,375.00	
		6,335.00
Library additions of books, periodicals, etc., binding of exchanges and stationery (expenditure from appropriation of $1,500),		1,381.07
Furniture and fixtures,		106.50
Balance,		7,350.91
		$66,301.33

(Signed) FRANK LYMAN, *Treasurer*.

NEW YORK, N. Y., January 23, 1913.

We have examined the above statement, compared it with the books and vouchers, and find same correct.

(Signed) BARROW, WADE, GUTHRIE & Co.
Certified Public Accountants.

PROCEEDINGS OF THE COUNCIL FOR THE YEAR 1912.

Meeting, Mar. 22, 1912.—The following Committee on Membership was appointed: Joseph Struthers, *Chairman;* B. B. Thayer, Karl Eilers, S. J. Jennings, A. L. Walker.

The following Special Committee on the Lord Kelvin Memorial Fund was appointed: James F. Kemp, *Chairman;* James Douglas, Samuel B. Christy, Joseph Struthers, *Secretary-Treasurer.* Authority was given to this Committee to join with other organizations, provided no expense would be incurred to be met by the Institute.

The following Special Committee to consider a joint meeting of the Institute with the Eighth International Congress of Applied Chemistry in New York, September, 1912, was appointed: Joseph W. Richards, *Chairman;* George C. Stone, William L. Saunders.

The By-Laws of the Boston Local Section of the American Institute of Mining Engineers were approved and ordered on file.

The following Special Committee to study the suggestion of affiliation between the American Institute of Mining Engineers and the Lake Superior Mining Institute to the extent of interchange of publications was appointed: J. Parke Channing, *Chairman;* James R. Finlay, William Kelly, Joseph W. Richards.

The following Special Committee to study the question of the formation of an Iron and Steel Section of the Institute was appointed: Charles F. Rand, *Chairman;* Joseph W. Richards, James Gayley, Bradley Stoughton, John Birkinbine, Richard Moldenke, Leonard Waldo, W. R. Webster.

Meeting, Apr. 26, 1912.—The new form of Proposal for Membership blank, recommended by the Committee on Membership, was approved.

The Special Committee on the Formation of an Iron and Steel Division of the Institute, Mr. Rand, *Chairman,* reported favorably, and on motion the following resolution was unanimously adopted:

Resolved, That there be established an Iron and Steel Division to represent the interests of the Institute in the Mining and Metallurgy of Iron and Steel; its business to be in charge of a Committee upon which shall rest the responsibility of se-

curing papers and discussions on Iron and Steel for the meetings of the Institute.

President Kemp subsequently appointed the following Committee: Charles Kirchhoff, *Chairman;* Charles F. Rand, *Vice-Chairman;* Bradley Stoughton, *Secretary;* James Gayley, John Birkinbine, Joseph W. Richards, Richard Moldenke, Leonard Waldo, William R. Webster, Henry M. Howe, Robert W. Hunt, Charles K. Leith, Julian Kennedy, F. W. C. Schniewind, Henry D. Hibbard, Felix A. Vogel.

The death of William A. Lathrop, Vice-President of the Council, was officially reported and a Committee of Two, consisting of the President and the Secretary, was appointed to prepare suitable resolutions. Mr. Robert V. Norris, of Wilkes-Barre, Pa., was elected a Vice-President to fill the vacancy caused by the death of Mr. Lathrop, the term of office expiring February, 1913.

Hennen Jennings was appointed a member of the Council to fill the place of Charles F. Rand, who had been elected to the Board of Directors.

The President announced the appointment of John H. Janeway, Jr., to fill the remaining vacancy on the Committee on Membership.

The application of the Mining Engineering Society of the University of Arizona for enrollment as an Affiliated Student Society was accepted.

Capt. A. F. Lucas was appointed, to serve with two other representatives to be appointed later, on a Special Committee of the Institute on Patent Law Legislation.

It was voted that the President appoint a Committee of Five members, not members of the Council, to send to the Secretary on or before Nov. 1, 1912, the names of consenting nominees for the elective offices next falling vacant under the Constitution.

Meeting, May 24, 1912.—It was voted that Joseph W. Richards be authorized to make the necessary time-arrangements with the Special Committee for holding the joint meeting with the Eighth Congress of Applied Chemistry on Sept. 6 and 7, and with the International Society for Testing Materials on Sept. 5, 1912.

A formal application for permission to establish the Puget Sound Local Section of the Institute was granted.

Meeting, June 28, 1912.—It was voted that the Secretary be authorized to proceed with the publication of the Emmons Volume on Ore-Deposits and to make such public announcements in the *Bulletin* and by circular concerning this work as may be deemed desirable.

It was voted that the Council recommend to the Board that the request of Mr. L. K. Armstrong, *Secretary*, be granted, "that the initiation fees of new members be applied to defray the expenses of the Spokane Local Section until said amount should reach 25 per cent. of the annual dues received from the membership enrolled in the Local Section." Said action to begin Jan. 1, 1912.

The following additional appointments to the Iron and Steel Division Committee were announced: Albert Sauveur, A. A. Stevenson.

The following Nominating Committee was appointed: Philip N. Moore, St. Louis, Mo., *Chairman;* John Birkinbine, Philadelphia, Pa.; D. W. Brunton, Denver, Colo.; Seeley W. Mudd, Los Angeles, Cal., Charles P. Perin, New York, N. Y.; Arthur L. Walker, New York, N. Y.

The following Special Committee on Method of Printing the Publications of the Institute was appointed: Joseph Struthers, Charles Kirchhoff, W. R. Ingalls.

Meeting, Sept. 27, 1912.—The Secretary announced the death, June 18, 1912, of M. Floris Osmond, an Honorary Member of the Institute. The following minute was unanimously adopted, and the Secretary was instructed to send a copy thereof to the relatives of M. Osmond:

"This Council has received with sorrow the tidings of the departure of M. Floris Osmond, an Honorary Member of the Institute, who died at his residence in Saint Leu, France, on June 18, 1912. Mr. Osmond's brilliant career as an original investigator and acute critic in the metallography of iron and steel had been recognized by all students and practitioners in Great Britain, Europe and America; and his contributions to the *Transactions* of this Institute commanded the admiration and gratitude of its members, expressed in his election to Honorary Membership in 1899. The Council shares in the

universal grief inspired by his too-early departure, in the prime of life, and desires to communicate to his friends and relatives, as well as to the professional world which mourns his death, its heartfelt sympathy."

The following Special Committee was appointed to present names for consideration for Honorary Membership in the Institute: James Douglas, *Chairman;* James F. Kemp, R. V. Norris.

The following Special Committee was appointed to study the question of the formation of a division of the Institute to cover the field of Precious and Base Metals: B. B. Thayer, *Chairman;* George C. Stone, Hennen Jennings, E. B. Kirby, T. B. Comstock.

The following delegates to represent the Institute at the Fourth National Conservation Congress at Indianapolis, Ind., Oct. 1 to 4, 1912, were appointed: J. A. Holmes and P. N. Moore.

Meeting, Oct. 4, 1912.—Prof. Richards, *Chairman* of the Special Committee of the Council and Board on Proposed Amendments to the Constitution to meet with the Committee of Five and report to the Council and Board, advised that a conference had been held; and on invitation the report of the joint meeting of the said Committees was presented by C. R. Corning and passed upon section by section, special note being made upon those sections on which an agreement could not be reached. The report was ordered on file.

It was voted that it was the sense of those members of the Council and Board present that as soon as possible the two governing bodies should be unified.

It was also voted that a Special Committee be appointed to study the question of unifying the two bodies and report to the Council and Board at a later meeting. The following Special Committee was appointed: Joseph W. Richards, *Chairman;* Charles F. Rand, and Charles Kirchhoff.

Meeting, Nov. 22, 1912.—The following appointments on the Committee on Patent Law Legislation, to serve with Capt. A. F. Lucas, were made: Gardner F. Williams, Hennen Jennings.

Meeting, Dec. 27, 1912.—Prof. Albert Sauveur, of Harvard University, was elected a representative of the Institute on the John Fritz Medal Board of Award for the term of January, 1913, to January, 1917, to succeed E. Gybbon Spilsbury, whose term expires January, 1913.

The appointment of the following representatives of the Institute to serve on a Special Committee of the American Institute of Electrical Engineers to aid in the deliberations on the use of electricity in mines was announced: Charles W. Goodale, William Kelly, Louis S. Noble.

The Secretary reported that all the material for the Emmons Volume on Ore-Deposits was in the hands of the printer, and that the book would probably be issued by Mar. 1, 1913. The price of $5 for cloth binding and $6 for half-morocco binding, as approved by the Finance Committee, was duly confirmed.

The Special Committee on Printing, Joseph Struthers, *Chairman*, reported that the question of printing was so closely involved with the policy to be pursued that it desired the Council to decide upon the policy before proceeding further with the work. The Council recommended the employment of a manufacturing publisher in a consulting capacity, and requested the Committee to present to the Council alternative policies with the corresponding estimates and cost of carrying out each.

The application of the Mining Society of the Pennsylvania State College for enrollment as an Affiliated Student Society of the Institute was granted.

Meeting, Jan. 23, 1913.—The following Special Committee on Precious and Base Metals was appointed: Charles W. Goodale, Butte, Mont., *Chairman;* ——— ———, *Secretary* (to be appointed by the *Chairman*); Theodore B. Comstock, Los Angeles, Cal.; Edmund B. Kirby, St. Louis, Mo.; George C. Stone, New York, N. Y.; Hennen Jennings, Washington, D. C.; David W. Brunton, Denver, Colo.; Leonard S. Austin, Salt Lake City, Utah; James MacNaughton, Calumet, Mich.; Thomas J. Grier, Lead, S. D.; L. D. Ricketts, Cananea, Mex.; Charles W. Merrill, San Francisco, Cal.; Willet G. Miller, Toronto, Can.; Samuel S. Fowler, Nelson, B. C.

The President reported that on account of disability John Birkinbine had resigned from the Nominating Committee and that he had appointed E. Gybbon Spilsbury to fill the vacancy thus created.

JOSEPH STRUTHERS, *Secretary*,

INSTITUTE ACTIVITIES FOR THE YEAR 1912.

Meetings of the Institute.

There were held, during the year 1912, two meetings of the Institute for the presentation and discussion of technical papers —the 102d meeting at New York City, Feb. 18–20, 1912, and the 103d meeting at Cleveland, Ohio, Oct. 29–31, 1912.

A detailed record of the proceedings of these meetings, including a description of the entertainments and excursions connected therewith, has been published and duly distributed to the membership—the New York meeting in *Bulletin* No. 63, March, 1912, pages 345 to 355; the Cleveland meeting in *Bulletin* No. 71, November, 1912, pages 1345 to 1355, and *Bulletin* No. 72, December, 1912, pages xii. to xiv.

The precedent was established of combining the annual business meeting of the Institute in February at New York with the technical sessions usually called in the spring or summer; so as to make the annual meeting the principal feature of interest during the year.

Local Sections.

During the year 1912 a local section has been formed at Puget Sound, which makes the total number 4, representing Boston, New York, Puget Sound, and Spokane, where meetings have been held as recorded regularly in the *Bulletin*.

Affiliated Student Societies.

Two organizations of undergraduates at technical schools and universities were recognized as Affiliated Student Societies during the year 1912. The total number of Societies enrolled is 23, representing the following institutions: Yale University—Sheffield Scientific School, University of Illinois, University of Nevada, University of Wisconsin, Lehigh University, University of Minnesota, Massachusetts Institute of Technology, University of Kansas, University of Idaho, State College of Washington, University of Texas, Ohio State University, Stanford University, Columbia University, University of California, Tufts College, University of Washington, Iowa State College, Missouri School of Mines, Case School of Applied Science, Colorado School of Mines, University of Arizona, and University of Kentucky.

Publications.

Transactions.—Vol. XLII. of the *Transactions*, an octavo of 1002 pages, comprising 51 papers and 6 discussions presented during the year 1911, was issued to the membership in June.

The proceedings, papers and discussions published by the Institute during the year 1912 were so many and so extensive that it was impracticable to include all in one regular volume without increasing the size beyond reasonable limits. It was therefore decided to issue two volumes for the publications of 1912—Vol. XLIII., to contain the papers and discussions of the New York Meeting in February, 1912, and of the meetings of the New York Local Section and the Spokane Local Section; and Volume XLIV., to contain certain papers and discussions of the joint meeting of the Institute with the Eighth International Congress of Applied Chemistry, New York, September, and of the Cleveland meeting, October, 1912. Volume XLIII., now nearly complete, is expected to be ready for distribution in April, 1913.

The Emmons Volume.—This volume is a continuation of the "Posepny Volume" and contains 29 important papers descriptive of ore-deposits and discussions of their origin, which have appeared in the *Transactions* subsequent to the publication of the former volume. The work is edited, with an introduction, by Dr. F. S. Emmons. It contains also a Biographical Notice of Dr. Emmons by his associate and friend, Dr. George F. Becker, and a comprehensive bibliographical Index of the Science of Ore-Deposits, prepared by Prof. John D. Irving, of the Sheffield Scientific School of Yale University, H. D. Smith, and H. G. Ferguson, concluding with a carefully prepared index. The volume will contain about 1000 pages, illustrated, and is expected to be off the press in April, 1913. The price has been set as follows:

Bound in cloth, $5.00. Bound in half-morocco, $6.00.

Bulletin.—Twelve numbers of the *Bulletin* (Nos. 61 to 72), containing the technical papers and discussions of the Institute (in "subject to revision" form) and announcements of general interest to the members of the Institute, such as Library accessions and requirements during the year 1912, notices of meetings of the Institute and other societies, lists of proposed members and associates, changes of addresses, deaths of members, obitu-

ary notices, etc., have been published and distributed promptly throughout the year 1912. The number of pages occupied by technical papers and discussions amounts to 1588 (as compared with 1042 pages during 1911), to which are to be added 341 pages of announcements, and 324 pages of advertising matter, making a total of 2253 (as compared with a total of 1802 pages during 1911) pages of printed matter.

Membership.

Changes in the membership of the Institute have taken place during the year 1912 as follows:

265 members and 23 associates have been elected; 42 members have been reinstated, 1 life associate has become a life member, and 8 associates have become members; a total of 339. Against this number should be recorded the deaths of 1 honorary member, 46 members, and 3 associates; the resignation of 77 members and 1 associate; and the dropping from the roll, by reason of non-payment of dues, of 108 members and 3 associates—a total of 239, which shows a net gain in the membership of 91. The net loss in membership in 1911 was 36. The total membership on Jan. 1, 1913, was 4,265.

The Library Committee.

James F. Kemp, *Chairman*, Charles Kirchhoff, and George C. Stone.

There were no stated meetings of the Library Committee during the year, but the Chairman or his representative attended the regular meetings of the Library Conference Committee of the three Founder Societies, called monthly.

The general management of the Library administration was discussed informally in connection with similar discussion by the Library Committees of the other Founder Societies.

The By-Laws of the United Engineering Society pertaining to the Library have been extensively amended so as to place the administration under a Board of Control, which shall be composed of four members appointed from each of the three Founder Societies, the Secretary of each of the Founder Societies, and the Librarian who shall act as Secretary of the Board—a total of 16.

These amendments were accepted by the Board of Trustees of the United Engineering Society.

Committee on Membership.

The following is the report of the Committee on Membership for the year ending Feb. 20, 1913. Dr. Joseph Struthers, *Chairman*, Karl Eilers, Benjamin B. Thayer, Sidney J. Jennings, and Arthur L. Walker were appointed on the Committee on Membership, Mar. 22, 1912. Professor Walker did not accept the appointment, and Mr. John H. Janeway, Jr., was appointed Apr. 26, 1912, to fill the vacancy.

During the year, the Committee met monthly, except in July and August, and acted upon 353 cases, and made the following recommendations to the Council: for election as members, 308; for election as associates, 35; for change of status from life associate to life member, 1; from change of status from associate to member, 6. Each case was carefully and earnestly considered and the Committee has endeavored to maintain the high grade of requirements for admission to the Institute.

Early in the year a new form of "Proposal for Membership" blank was prepared by the Committee, requesting specific information concerning the professional and social qualifications of the candidate in order that the Committee could form an intelligent opinion. This form was later approved by the Council.

MEMBERSHIP.

The following list comprises the names of those persons elected as members, who duly accepted election during the year 1912. The marks used to designate the different classes of membership are: Life Member, **; Member,*; Associate Member, †. Heavy-faced type signifies Honorary Membership.

*Abbott, Robert R., Cleveland, Ohio.
*Adami, Charles J., Rivermines, Mo.
*Alden, Herbert W., Detroit, Mich.
*Ammon, Mark A., Cleveland, Ohio.
*Arnold, Ralph, Los Angeles, Cal.
*Barager, George W., Lattimer Mines, Pa.
*Barlow, Alfred E., Westmont, Que., Can.
*Barnard, Enoch A., Anaconda, Mont.
*Barnes, Walter A., Tooele, Utah.
†Barnum, W. E., Pachuca, Hidalgo, Mex.
*Barratt, S. A., Santiago de Cuba, Cuba.
*Baxter, Charles H., Loretto, Mich.
*Beck, Erich A., New York, N. Y.
*Bedford, Robert H., Grass Valley, Cal.
*Bell, Wm. L., Greenwood, B. C., Can.
*Bennett, Arthur F., New York, N. Y.
*Bent, Quincy, Sparrows Point, Md.
*Best, William N., New York, N. Y.
*Bonnyman, James, Birmingham, Ala.
*Borowsky, Abraham G., Atco, N. J.
*Braman, Harry S., Youngstown, Ohio.
*Broden, Albert, Reading, Pa.
*Brooks, John Y., Cleveland, Ohio.
*Brown, Holcombe J., Buffalo, N. Y.
**Browning, W. C., Salt Lake City, Utah.
*Bull, Robert A., Granite City, Ill.
*Burcham, William D., Shafter, Tex.
*Burley, Curtis R., Gem, Ida.
*Callaway, Fred W., Kellogg, Ida.
*Cantley, C. L., New Glasgow, Nova Sco.
*Carney, Frank D., Steelton, Pa.
*Chamberlain, M., Chattanooga, Tenn.
*Clagett, Thomas H., Bluefields, W. Va.
*Clapp, Laurence R., Silverton, Colo.
*Cooke, Fred W., Jr., New York, N. Y.
*Cornell, R. L. G., Kanwa, Congo Belge, W. Africa.
*Cottrell, Fred G., San Francisco, Cal.
*Cousin, Robert, London, England.
*Coyle, John A., Bridgeton, N. J.
*Crandall, Roderic, Palo Alto, Cal.
*Crankshaw, Hugh M., Lansford, Pa.
*Crook, W. J., Stanford University, Cal.
†Crowther, John S., Jr., Toledo, O.
*Cunniff, Bernard, Chicago, Ill.
*Dake, Walter M., Jr., Rye Valley, Ore.
*Dale, William C., Copiapo, Chile.
*Dalton, Michael P., Mullan, Ida.
*Davis, Walter W., Leadville, Colo.
*Davis, William W., Steelton, Pa.
*Dickerman, Wm. C., New York, N. Y.
*Doak, Samuel E., Glen Wilton, Va.
*Dowler, Harry P., Heilwood, Pa.
*Dows, David, New York, N. Y.
*Earle, Theodore, New York, N. Y.
†Elkan, Benno, New York, N. Y.
*Elliott, Roy H., Telluride, Colo.
*Emerson, E. H., Santiago de Cuba, Cuba.
†Eppley, Marion, New York, N. Y.
†Eshleman, S. K., Jr., Leaman Pl., Pa.
*Farrant, James C., New York, N. Y.
*Ferguson, Claude, Caldwell, Ida.
*Flagg, Arthur L., Spokane, Wash.
**Frechette, Howells, Ottawa, Ont., Can.
*Friedlander, O. A., Obuasi, Gold Coast Colony, W. Africa.
*Fulton, John A., Melones, Cal.
*Furniss, Thomas A., Punxsutawney, Pa.
*Fyfe, Alex., Ventanas, Durango, Mex.
*Galen, C. F., Rioverde, S. L. P., Mex.
*Gannon, M. H., Port Hill, Ida.
*Gardner, E. D., Missoula, Mont.
*Garnier, Thomas R., S. Pasadena, Cal.
*Gerhardt, Reginald B., Harrisburg, Pa.
*Gibbs, Geo. H., Briton Ferry, S. Wales.
*Gmehling, Andreas, Santiago, Chile.
*Goedicke, Carl, San Antonio, Tex.
*Goetter, F. B., San Dimas, Durango, Mex.
*Goltra, Edward F., St. Louis, Mo.
*Goodrich, Harold B., New York, N. Y.
*Grace, Eugene G., S. Bethlehem, Pa.
†Graef, Harry C., New York, N. Y.
†Graybill, John H., Pittsburg, Pa.
*Greenough, Warren E., Wallace, Ida.
†Greensfelder, N. S., Chewelah, Wash.
*Gwinn, James W., Kellogg, Ida.

*Hall, John H., New York, N. Y.
*Hall, William J., Wallace, Ida.
**Hamilton, J. W. H., New York, N.Y.
*Hartman, William E., Chicago, Ill.
*Hendricks, Geo. F., Cobalt, Ont., Can.
*Hendrickson, Wm. H., Frisco, Utah.
*Hioki, Masaaki, Akita, Japan.
†Hochschild, Berthold, New York, N.Y.
*Hofecker, C. A., Constantinople, Turkey.
*Hoffmann, Karl F., Mexico City, Mex.
*Hofstrand, Oscar B., Wallace, Ida.
*Holden, J. O. E., Princeton, B. C., Can.
*Holderer, Geo. B., New York, N. Y.
†Holmes, Richard G., Spokane, Wash.
*Hoover, William M., Hazelton, Pa.
*Houser, John N., Carterville, Mo.
*Hunnings, S. V., Schenectady, N. Y.
*Hurd, W. E., Haileybury, Ont., Can.
**Hutchinson, E. C., San Francisco, Cal.
**Inouyé, Tadashiro, Tokyo, Japan.
*Ingalsbe, Frank R., Missoula, Mont.
*James, Ulysses S., Newark, N. J.
*Janin, Charles, San Francisco, Cal.
*Jayne, Amel R., Seattle, Wash.
†Johnston, Roderick L., Chrome, N. J.
*Jones, Archie R., Tonopah, Nev.
*Jones, Lewis M., Pittsburg, Pa.
†Jones, Zechariah, Republic, Wash.
*Joralemon, Ira B., Warren, Ariz.
*Jordan, Harold S., Campo Seco, Cal.
*Jorgensen, Frank F., Buxton, Iowa.
*Jui, Pao-vung, Shanghai, China.
*Kervin, George A., Kennett, Cal.
*Kidder, Sidney J., Blair, Nev.
*Kinney, Harry D., Tooele, Utah.
*Kirkland, T. C., Fundicion, Son., Mex.
*Klepinger, J. H., Great Falls, Mont.
*Klopstock, Paul, Kennedy, Nev.
*Kuzell, Charles R., Great Falls, Mont.
*Lamb, Wm. H., Jr, Birmingham, Ala.
*Landgrebe, Karl L., Ensley, Ala.
*Laucks, Irving F., Seattle, Wash.
*Leaning, Eugene H., Scranton, Pa.
*Leland, Everard, Gleeson, Ariz.
*Libby, Fay W., Wickenburg, Ariz.
*Limbach, Edmund C., Loomis, Wash.
*Lindau, Sam P., Swansea, Ariz.
*Linhardt, Harry W., Columbus, Ohio.
*Lipscomb, Robert A., Low Moor, Va.
†Loeb, Carl M., New York, N. Y.
*Logan, John W., Conshohocken, Pa.
*Longan, Walker B., Breckenridge, Colo.
†Lonsbery, Geo. A., Los Angeles, Cal.
*Loring, Edward A., London, England.
*Lower, Joseph B., Unsan, Korea.
**McCreery, James H., New York, N.Y.
*McGinnis, R. B, San Francisco, Cal.
*McIntyre, John E., San Antonio de la Huerta, Son., Mex.
*McIntire, Robert, Radersburg, Mont.
*McKay, Richard V., Steelton, Pa.
*McKee, Arthur G., Cleveland, Ohio.
*McKee, Wm. C., Indiana Harbor, Ind.
*McKinnis, Charles, Wallace, Ida.
*McLeod, Howard D., Seattle, Wash.
*McNair, Fred W., Houghton, Mich.
*Maben, J. C., Jr., Birmingham, Ala.
*MacCarthy, Marion S., Denver, Colo.
*Macfee, Robert, Batum, S. Russia.
*Madge, William C., London, England.
*Manion, Edward, Lead, S. D.
*Marsh, H. W., White Horse, Y.T., Can.
†Marshall, Emory M., Mason, Nev.
†Martin, Samuel G., Eureka, Colo.
*Mattievich, Emerio, Batum, S. Russia.
*Maynard, Edward W., Senter, Mich.
*Merrett, W.H., Wallington, Surrey, Eng.
*Merrill, F. J. H., Los Angeles, Cal.
*Metcalf, Frank A., Wardner, Ida.
*Meyers, William R., Princeton, Mich.
*Middlekamp, Leroy L., Ellamar, Alas.
*Millard, H. Alfred, New York, N. Y.
*Millard, Robert P., Cleveland, Ohio.
*Millard, William J., New York, N.Y.
*Milliken, J. T., Colo. Springs, Colo.
*Mishler, Ralph T., Ysabal, Son., Mex.
*Mooney, James D., Butte, Mont.
*Morgan, Charles G., Roslyn, Wash.
*Morgan, George H., Los Angeles, Cal.
*Morrison, Harold A., Candor, N. C.
*Morrow, Bayard S., Gem, Ida.
*Morse, George H., Pocahontas, Can.
*Munroe, Harold S., Butte, Mont.
†Murphy, Edward M., Burke, Ida.
*Newbaker, Edward J., Audenried, Pa.
*Newhall, H. B., Jr., New York, N. Y.
*Newton, Harry W., Republic, Wash.
*Notman, Arthur, Bisbee, Ariz.
*Oberg, Anton C., Hibbing, Minn.
*O'Connor, Edward S., Mayville, Wis.
*Orr, John F., New York, N. Y.
*Page, Edwin R., Jodie, W. Va.
*Paine, Francis W., Houghton, Mich.
*Palmer, C. H., Jr., Las Vegas, Nev.

*Parrock, H. P., Buffalo, N. Y.
*Parsons, Roscoe R. S., Bonne Terre, Mo.
*Paterson, Arthur W., Lewistown, Ida.
*Patterson, Seeley B., Jr., Douglas, Ariz.
†Pearce, Samuel L., Los Angeles, Cal.
*Peattie, Clarence G., Oxford, N. J.
*Perry, William C., Cumberland, Md.
*Phalen, William C., Washington, D.C.
*Portugal, Joseph H., Keeler, Cal.
*Pratt, Paul D., Libby, Mont.
*Preston, Veryl, New York, N. Y.
*Prickett, William C., Birmingham, Ala.
*Prommel, H. W. C., Mexico City, Mex.
*Randall, Charles A., Cobalt, Ont.
*Raymer, George S., Cambridge, Mass.
*Reed, Alfred C., Port Orange, Fla.
*Reynders, John Van W., Steelton, Pa.
*Rightmire, R. E., Fairmont, W. Va.
*Rio, José P., Bahia, Brazil.
*Roddewig, George W., Larson, Ida.
*Roden, Benjamin F., Marvel, Ala.
*Rodgers, C. E., Sheep Creek, B.C., Can.
†Roeser, E. B., Kanawha Falls, W. Va.
*Rood, V. S., Bingham Canyon, Utah.
*Rowe, Fremont S., Burke, Ida.
*Ryall, George M., New York, N. Y.
*Salas, Luis E., New York, N. Y.
*Sargent, FitzWilliam, Mahwah, N. J.
*de Saulles, C. A. H., New York, N. Y.
*Saunders, T. S., Aire Libre, Puebla, Mexico.
*Schiffner, Otto E., Dale, Cal.
*Schneider, George W., Golden, Colo.
*Schofield, Walter D., Trinidad, Colo.
*Schofield, Wm. McN., Eckman, W. Va.
*Schuettenhelm, J. B., Garfield, Utah.
*Schumacher, Wilhelm, Berlin, Ger'ny.
*Sedgwick, A. E., Mexico City, Mex.
†Sethna, Nanabhoy R., New York, N. Y.
*Severy, Clarence L., Pasadena, Cal.
*Shimer, Wm. R., Bethlehem, Pa.
*Siebenthal, C. E., Washington, D. C.
*Siebenthal, W. A., Republic, Mich.
**Seibert, Percy A., La Paz, Bolivia.
*Sinclair, M .J., Kellogg, Ida.
*Singewald, J. T., Jr., Baltimore, Md.
*Slosson, H. L., Jr., San Francisco, Cal.
*Smith, James W., Trenton, N. J.
*Smith, R. E., Vladivostok, Siberia.
*Smith, Sydney, Juneau, Alaska.
*Smith, Webb, Martell, Cal.
*Snyder, Henry S., So. Bethlehem, Pa.
*Sparks, James T., Baldy, N. M.
*Stakel, Charles J., Ishpeming, Mich.
*Stephen, Walter M., Seattle, Wash.
*Stevens, John V., Coquimbo, Chile.
*Stillman, Charles A., So. Chicago, Ill.
*Strohecker, J. W., Rossland, B.C., Can.
*Tate, E. L., Spokane, Wash.
**Tatton, Edwin H., Chocaya, Bolivia.
*Thomas, Perry C., Macdonald, W. Va.
*Thompson, C. H., Darbyville, Va.
*Thompson, Frank E., Johnstown, Pa.
*Tinsley, R. B., Corozal, Canal Zone, Pan.
*Toy, Francis L., Pittsburg, Pa.
*Tranerman, Carl J., Tuscarora, Nev.
*Treat, L. B., Guatemala City, Guatem.
*Trenerry, E. J., Moonta Mines, S. Aus.
†Trotter, Felix F., Corey, Ala.
*Turner, Charles P., Steelton, Pa.
*Turner, Wm. V., Rancagua, Chile.
*Van Siclen, M., Turtle Creek Siding, N. B., Canada.
*Van Valkenburg, R. D., Chicago, Ill.
*Van Winkle, C. T., Salt Lake City, Utah.
*Waeber, G. R., Iron River, Mich.
*Wallower, Frank C., Webb City, Mo.
**Watanabe, Wataru, Tokyo, Japan.
*Weigall, Henry S., Doten, Korea.
†Westlake, Emory H., New York, N. Y.
*Wheelwright, O. W., Florence, Wis.
*White, E. E., Glen White, W. Va.
*Whiting, Lowe, Iron River, Mich.
*Wiggin, Albert E., Anaconda, Mont.
*Wilfley, Clifford R., Ouray, Colo.
*Wilkins, Llewellyn H., London, Eng.
*Willard, Lewis L., Uniontown, Pa.
*Williams, J. P., Jr., Tazewell, Va.
*Wilson, Arden M., Telluride, Colo.
*Wilson, Fred C., Felton, Cuba.
*Wilson, John B., Wallace, Idaho.
*Wolf, Otto C., Philadelphia, Pa.
*Wood, Frederick W., Baltimore, Md.
*Wraight, E. A., Streatham Hill, Eng.
*Wright, C. A., Washington, D. C.
*Yamada, Naoya, Tokyo, Japan.
†Yang, Cho, So. Bethlehem, Pa.
*Yeager, Howard M., Pineville, Ky.
*Youd, Herbert, Guarda, Portugal.
*Young, Charles D., Altoona, Pa.
*Young, H. W., Stanford Univ., Cal.
*Yungbluth, A. J., Ishpeming, Mich.
*Zalinski, E. R., Salt Lake City, Utah.

Deaths.

The following list comprises the names of members whose deaths have been reported to the Secretary of the Institute during the year 1912:

Date of Election.	Name.	Date of Decease.
1875.	**Attwood, George,	———.
1892.	*Bachman, D. I.,	Oct. 8, 1912.
1902.	**Buckley, E. R.,	Jan. 19, 1912.
1903.	**Carter, Thos. L.,	Sept. 2, 1912.
1895.	**Chalmers, J. A.,	Sept. 9, 1912.
1890.	*Chisholm, John,	May 12, 1912.
1897.	†Church, M. J.,	May 21, 1912.
1880.	**Cliff, John,	May 24, 1912.
1883.	*Coryell, Torbert,	Apr. 18, 1912.
1887.	**Dougherty, C. E.,	May 24, 1912.
1907.	*Eatherly, Adrian D.,	———.
1890.	*Farrel, Franklin,	Jan. 11, 1912.
1897.	*Fleming, J. B.,	Dec. 17, 1911.
1889.	*Grant, James B.,	Nov. 1, 1911.
1891.	*Hall, Harry R.,	Dec. 11, 1912.
1892.	*Hartzell, H. K.,	———.
1892.	*Hobson, John B.,	Jan. 9, 1912.
1890.	*Hofius, W. D.,	Feb. 27, 1912.
1894.	*Jackson, T. Moore,	Feb. 3, 1912.
1906.	*Kane, Daniel B.,	Jan. 3, 1912.
1872.	*Kent, Joseph C.,	Aug. 31, 1912.
1882.	*King, Thomas M.,	Sept. 13, 1911.
1909.	*Kirk, Harry E.,	July 31, 1912.
1883.	*Lathrop, W. A.,	Apr. 12, 1912.
1889.	*McCrery, Charles,	Sept.—, 1912.
1904.	*McLoughlin, H. M.	July 30. 1912.
1906.	*McLoughlin, W. L.,	———.
1905.	*Maffett, R. R.,	Apr. 9, 1912.
1877.	*Meister, H. C.,	Nov. 27, 1911.
1898.	**Morris, James,	———.
1900.	*Mussen, H. W.,	Apr. 18, 1912.
1871.	†Oliver, Gen. P. A.,	May 18, 1912.
1899.	*Osmond, F. (Hon).	June 27, 1912.
1902.	**Poindexter, C. L.,	July 29, 1912.
1881.	**Pullman, J. W.,	May 5, 1912.
1910.	*Sampson, R. E.,	Dec. 12, 1912.
1903.	*Saylor, B. F. A.,	Dec. 2, 1911.
1893.	*Sjöstedt, E. A.,	Apr. 15, 1912.
1906.	*Smart, Geo. O.,	Nov. 29, 1911.
1871.	*Smith, T. G.,	Feb. 20, 1912.
1899.	*Snedaker, J. A.,	———, 1912.
1871.	**Squire, Joseph,	Oct. 24, 1911.
1905.	*Stevens, H. J.,	Apr. 22, 1912.
1881.	**Taylor, Percyvale,	May 24, 1912.
1891.	*Touzeau, E. M.,	Nov. 15, 1911.
1904.	*Wainwright, J. H.,	Dec. 29, 1911.
1902.	*Wanless, Geo. J.,	Mar. 3, 1912.
1911.	†Weaver, H. M.,	Oct. 3, 1912.
1886.	*White, Maunsel,	Oct. —, 1912.
1894.	*Young, John W.,	Feb. 14, 1912.

* Member. ** Life Member. † Associate.

Proceedings of the One Hundred and Second Meeting, New York, N. Y., February, 1912.

COMMITTEES.

Reception, Monday, Feb. 19.—Benjamin B. Lawrence, *Chairman*.

W. A. Bostwick,
W. DeL. Benedict,
J. Parke Channing,
W. B. Kunhardt,
A. R. Ledoux,
Edwin Ludlow,
Charles F. McKenna,
Ambrose Monell,
H. S. Munroe,
C. P. Perin,
Pope Yeatman.
Charles Rand,
D. M. Riordan,
P. de P. Ricketts,
Thomas Robins,
E. G. Spilsbury,

Reception, Tuesday, Feb. 20.—Arthur S. Dwight, *Chairman*.

Lawrence Addicks,
J. H. Banks,
William Campbell,
A. C. Carson,
B. E. Eldred,
H. W. Hardinge,
L D. Huntoon,
J. H. Janeway, Jr.,
W. McA. Johnson,
C. H. Jouet,
E. L. Kurtz,
J. F. McClelland,
R. G. Moldenke,
W. H. Nichols, Jr.,
H. A. J. Wilkens.
Charles Of,
H. A. Prosser,
E. Randolph,
G. C. Stone,
F. W. C. Schniewind,
Leonard Waldo,
A. L. Walker,

Reception, Wednesday, Feb. 21.—John D. Irving, *Chairman*.

A. Chester Beatty,
A. A. Blow,
William Braden,
J. Morgan Clements,
R. A. Cook,
E. L. Dufourcq,
Theodore Dwight,
L. W. Francis,
E. O. Hovey,
Sidney J. Jennings,
Joseph W. Roe,
W. L. Saunders,
J. M. Sherrerd,
F. M. Simonds.
H. Souther,
E. S. Sperry,
J. R. Stanton,
Knox Taylor,
H. S. Washington,
J. E. Woodman.

Dinner, Feb. 20.—James Gayley, *Chairman*.

Albert R. Ledoux,
D. M. Riordan,
Theodore Dwight,
Thomas Robins.

Finance.—George D. Barron, *Chairman*; Theodore Dwight.

The 102d meeting of the Institute was held at the Institute headquarters in the Engineering Societies Building, New York, N. Y., on Feb. 19, 20 and 21, 1912. A Bureau of Information, in charge of Mr. George Buckman, was maintained during the meeting.

The proceedings opened with an informal social meeting on Monday evening, Feb. 19, in Assembly-Room No. 1, on the

fifth floor of the building. The meeting was in charge of the New York Section of the Institute, and Mr. Benjamin B. Lawrence, Chairman of the Reception Committee for the day, delivered the address of welcome, after which Mr. Charles Kirchhoff, President of the Institute, addressed the gathering on Institute Aims and Ideals, as follows:

During the past few years there have been far-reaching and notable developments in the work of many of the national and local technical societies. The Carnegie gift has stimulated enthusiasm, and has fostered co-operation among engineers through the completion of the Engineering Building on the professional side, and through the enlargement of the Engineers' Club on the social side. While the splendid results thus obtained have been most noticeable locally, their influence is penetrating slowly throughout the country, and combined efforts are being made to make them available in an increased measure to all. In these aims the Institute is actively participating.

I have observed that with wider opportunities and enlarged facilities, the ambitions of the national engineering societies have grown. They are beginning to go beyond the collection and distribution of technical data; they are seeking to directly promote progress through research committees, backed, if necessary, by adequate appropriations. Some magnificent work has been done in this direction abroad, a striking example being the study of the power-requirements of rolling-mills, by a commission of the Verein deutscher Eisenhuettenleute, under the direction of Dr. Puppe.

There are elements in every national engineering society which affect to a varying degree the attainment of its principal objects, which are the accumulation of facts and thoughts, the exchange and discussion of experience and practice; the fostering of acquaintance and friendship, and the promotion of the solidarity of the profession. These elements are the tendency towards specialization, and the scattering of the membership geographically. So far as both tendencies are concerned, the Institute is affected perhaps more than any other society. Specialization in mining and metallurgy, and the cognate sciences, has gone very far since the days of the organization of the Institute, and has had its deep effect upon it. That part of our membership which is directly connected with operating mines, mills, and works, is very much scattered in the nature of things.

To a greater or less extent, the principal technical societies have endeavored to meet these conditions in two ways, by the organization of Local Sections, and by the establishment of professional divisions. Particularly in the mining and metallurgical industries, local sections bring together members who have identical or closely allied professional interests, and a keen appreciation of the opportunities and advantages of social intercourse. This is true notably where the distractions of a large city are missing. During the past year a movement in the direction of the organization of local sections has been initiated, and while it has had a measure of success, there are often special conditions which must be worked out. The experience of other societies which have gone further in this direction would indicate that such organizations often need financial support, which in the aggregate involves considerable demands upon the resources of the parent body. But they should prove a powerful factor for mutual advantage. They offer to the active member of a local section the opportunity to obtain for his professional work an international audience where his paper has been accepted for publication in

the Transactions of the Society, which local engineering bodies cannot offer. They stimulate the presentation of information on practice in one field which is valuable to those in other sections and in other branches, and give the younger men opportunities for training in collecting and presenting facts and meeting comment and discussion.

The feeling that the tendency towards specialization must be reflected in some modification in the organization of the societies has existed for a long time, and has in one conspicuous instance led to the formation of a relatively large series of topical divisions.

Meetings at which a series of papers covering many parts of the field are brought up in succession can rarely be of sustained interest to more than a few of those in attendance. If the number of those present qualified to discuss or supplement a paper on a special subject is small, the author feels himself chilled by an indifferent or unsympathetic audience. In the Institute effort has been repeatedly made—taking advantage of the fact that the meeting was being held in a locality where particular industries predominated—to bring out papers connected with those industries, and induce attendance of members identified with them. Such efforts have been crowned with a measure of success, but they could not then be persistently followed up. I believe the time has come when they should be systematized by providing the facilities for the organization of topical sections. The mission of these would be the intense cultivation of their respective fields, in the place of the casual representation of different branches of the whole range of mining and metallurgy, to which the present methods cannot help being restricted. There must be a concentration of effort which shall bring to the members interested in each division the conviction that the Institute is not alone keeping abreast of progress in that field, but is leading it.

In an able address, delivered at the annual meeting in 1889, by the President of the Institute, Prof. W. B. Potter, he suggested for a grouping the following: I. Iron and Steel; II. The Precious and Base Metals; III Geology and Mining; IV. Chemistry. Probably we might to-day be inclined to split the third group into Economic Geology and Metal- and Coal-Mining, but it is much more strikingly true to-day than it was in 1889 that we must adjust the organization of the Institute to the changes in the relations of the individual members to the work which it is to perform. The Institute has a splendid mission before it in this direction, but if it does not rise to that mission, the inevitable result will be that a series of societies will be formed which will aim to carry out the work. From the standpoint of efficiency such an outcome should be deeply deplored. The advantages of one powerful organization in charge of interests which after all are interlocked at so many points, with the prestige and success of forty years of splendid achievements, should not be lightly surrendered to a series of bodies, each singly weak.

While doing more intense work in special fields, the Institute has missions in other directions which would be common to all interests, and would tie them together. Besides, its central organization could more effectively and cheaply carry on the purely administrative functions, and the editorial and publication work.

While the Institute could depend upon the voluntary efforts of many members now leaders in the different groups or sections, it would probably soon develop that the services of special officers, fairly compensated, would be required. The volume of the publications would be greatly enlarged, but the accessions to the membership at large, or to affiliates of special groups, would meet all additional outlays after a probable period of development. One leading society has gone so far as to print three separate sets of publications.

Another field of endeavor should be given close consideration, which the

founders of the Institute put forward with striking emphasis. Let me quote from the first circular sent out by Eckley B. Coxe, R. P. Rothwell, and Martin Coryell, dated Wilkes-Barre, April, 1871, which led to the organization of the Institute at the meeting of May 16, 1871:

"The great development of the mines and metallurgical work of this country during the last few years, accompanied as it has been by the investment of enormous sums of money in purchasing lands and in the erection of improvements, requires that advantage should be taken of the accumulated knowledge of engineers, superintendents and others in mastering the problems which are constantly presenting themselves for our action. Among those may be mentioned, the consideration of more economical systems of mining in our coal and metalliferous mines—improved methods of transportation above and below ground; unwatering and ventilating mines; the mechanical preparation of coal and other minerals; the various metallurgical processes, and in fact, every question tending to the attainment of the two great objects:

"1st. The more economical production of the useful minerals and metals.

"2d. The greater safety and welfare of those employed in these industries."

It will be observed that the signers of the first invitation laid particular stress upon the duty to safeguard the men employed in the mining and metallurgical industries; a mission of the Institute which has not received the attention which it deserved. It has, however, engaged public interest keenly in recent years, and splendid progress has been made by governmental and private agencies. These should have the active co-operation of the Institute, notably in the direction of discussing the many questions of a strictly technical nature, with which the members are so eminently qualified to deal.

This leads to the question, to what extent the Institute may and should participate in the discussion of public questions generally. Our Constitution provides that special committees may be appointed to make investigations and reports for presentation to the Institute, but specifically forbids that any action shall be taken binding the Institute for or against the conclusions embodied in any such reports. While that prohibition is beyond a doubt a very wise one, and avoids serious dangers, it would seem that a powerful influence for good may be exerted by committees' reports, and by the discussions to which they might lead.

The past achievements of the Institute justify the expectation that the highest estimate of its mission may be attained if the membership will co-operate heartily in its work. It is only by general participation in its enlarged activities that there can be realized the splendid possibilities for good to the profession, to the industries which it represents, and to the individual members themselves.

Prof. James F. Kemp related some humorous experiences in mountain-climbing as a prelude to his illustrated discourse on The Formation of Mineral Veins,* which terminated the program. The members then adjourned to the Institute rooms on the ninth floor of the building for further social intercourse.

The Annual Business Meeting of the Institute was held on Tuesday morning, Feb. 20, 1912.

The technical sessions were held in Assembly-Room No. 1, on the fifth floor of the Engineering Societies Building. Mr. Charles Kirchhoff presided at all of the sessions. The first

* Not furnished for publication.

session, Tuesday, Feb. 20, at 2 p.m., was allotted to papers on the sintering and briquetting of fine ores, flue-dust, etc. The following papers were presented in oral abstract by the authors, with the exceptions noted:

*The Sintering of Fine Iron-Bearing Materials by the Dwight & Lloyd Process, by B. G. Klugh, Birdsboro, Pa. (Read by George W. Maynard. Illustrated by samples of sintered material. Discussed by J. L. W. Birkinbine, Alfred H. Cowles, H. M. Chance, Arthur S. Dwight, Anton Eilers, Henry M. Howe, N. S. Keith, Charles Kirchhoff, George W. Maynard, Joseph W. Richards, F. W. C. Schniewind, E. Gybbon Spilsbury, J. W. Tudor, and F. A. Vogel.)

*Agglomeration of Fine Materials, by W. S. Landis, South Bethlehem, Pa. (Read by Joseph W. Richards.)

*Sintering and Briquetting of Flue-Dust, by Felix A. Vogel, New York, N. Y. (Mr. Vogel displayed a collection of briquettes and nodulized materials. The paper was discussed by Arthur S. Dwight, F. W. C. Schniewind, Otto Sussmann, Felix A. Vogel, and Arthur L. Walker.)

*The Schumacher Briquetting Process, by Joseph W. Richards, South Bethlehem, Pa. (Discussed by Alfred H. Cowles, Arthur S. Dwight, Henry M. Howe, Joseph W. Richards, and F. W. C. Schniewind.

The Briquetting of Iron-Ores, by N. V. Hansell, New York, N. Y. (Discussed by Alfred H. Cowles, N. V. Hansell, and George W. Maynard.)

*Electrostatic Concentration or Separation of Ores, by H. A. Wentworth, Boston, Mass. (Discussed by Robert H. Richards.)

The second session was held on Wednesday, Feb. 21, at 10 a.m.

The following papers were presented in oral abstract by the authors:

The James Diagonal-Plane Slimer, by S. Arthur Krom, New York, N. Y. (Illustrated by lantern-slides. Discussed by Robert H. Richards.)

*Progress in Roll-Crushing, by C. Q. Payne, New York, N. Y. (A pair of Frazee crushing-rolls showing the even effect of wear was exhibited. The paper was discussed by C. Q. Payne, Robert H. Richards, and E. Gybbon Spilsbury.)

* Pamphlet copies distributed.

Gold Hill Mining-District in Western Utah, by James F. Kemp, New York, N. Y.† (Illustrated by lantern-slides.)

Geology of the New Catskill Aqueduct, by Charles P. Berkey, New York, N. Y.† (Illustrated by lantern-slides.)

The Geology of Certain Salt-Deposits in Western Germany, by F. F. Hahn, New York, N. Y.†

The third technical session was held on Wednesday, Feb. 21, at 2 p. m. The following papers were presented in oral abstract by the authors, with the exceptions noted:

The Refining of Blister-Copper, by Horace H. Emrich, Kyshtim, Russia. (Read by R. W. Raymond. Discussed by Anton Eilers, N. S. Keith, and Albert R. Ledoux.)

*Geology and Ore-Deposits of the Silverbell Mining-District, Arizona, by C. A. Stewart, University of Idaho, Moscow, Idaho. (Read by James F. Kemp.)

Stagnant Mine-Waters, by Alfred C. Lane, Tufts College, Boston, Mass.†

The Decomposition of Metallic Sulphates at Elevated Temperatures in a Current of Dry Air, by H. O. Hofman and W. Wanjukow, Massachusetts Institute of Technology, Boston, Mass. (Read by H. O. Hofman. Discussed by H. O. Hofman, W. McA. Johnson, and Albert R. Ledoux.

*A Concise Method of Showing Ore-Reserves, by N. H. Emmons, Knoxville, Tenn. (Read by J. D. Irving. Discussed by John D. Irving, N. S. Keith, William Kelly, E. W. King, Charles Kirchhoff, and Alfred C. Lane.

*Electrical Fume-Precipitation, by F. G. Cottrell, San Francisco, Cal. (Illustrated by lantern-slides. Discussed by F. G. Cottrell, A. H. Elliott, L. C. Graton, F. W. C. Schniewind, and E. Gybbon Spilsbury.)

A new portable electric mine-lamp and a few photographs showing it in use as a miner's head-light and as a hand-lantern were exhibited at the close of the session. These exhibits were sent by David B. Rushmore to illustrate his paper, A New Electric Miners' Lamp.

The following papers were read by title for future publication by the Institute:

Vanadium in Pig-Iron, by Porter W. Shimer, Easton, Pa.

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* Pamphlet copies distributed.

† Not furnished for publication.

Temperature Conversion-Tables, by Leonard Waldo, New York, N. Y.†

*Treatment of Mine-Water from the Ashio Copper-Mine, by Joseph W. Richards, South Bethlehem, Pa.

*Rational Valuation and Quality-Efficiency of Furnace-Stock, by John Jermain Porter, University of Cincinnati, Cincinnati, Ohio.

Bearing of the Theories of the Origin of Magnetic Iron-Ores on Their Possible Extent, by Frank L. Nason, West Haven, Conn.

Geology of Harrison Gulch in Shasta County, Cal., by H. E. Kramm, Ithaca, N. Y.

*Direct Determination of Small Amounts of Platinum in Ores and Bullion, by F. P. Dewey, Washington, D. C.

An Early Discovery of Fullers' Earth in Arkansas, by J. C. Branner, Stanford University, Cal.

Occurrence of Silver-, Copper- and Lead-Ores at the Veta Rica Mine, Sierra Mojada, Coahuila, Mexico, by Frank R. Van Horn, Cleveland, Ohio.

Contribution to the Study of the Pre-Cambrian Rocks of the Harney Peak District of South Dakota, by Gordon S. Duncan, London, England.

Treatment of Complex Silver-Ore at the Lucky-Tiger Mine, El Tigre, Sonora, Mexico, by D. L. H. Forbes, Toronto, Ont., Canada.

The Solubility in Nitric Acid of Gold Contained in Certain Copper-Alloys (Copper-Bullions), by Edward Keller, Perth Amboy, N. J.

Fuel-Economy of Dry-Blast, by R. S. Moore, New York, N. Y.‡

The San Nicolás Mining-District, San Nicolás, Tamaulipas, Mexico, by I. H. Wentworth, Matehuala, Mexico.

Alaska Coal-Land Problems, by H. Foster Bain, San Francisco, Cal.

*Abrasion and Dust-Losses in Ore-Drying, by Carl F. Dietz and Dyke V. Keedy, Boston, Mass.

The Smokeless Coal-Field of West Virginia, by Edwin Ludlow, Eccles, W. Va.‡

A New Electric Miners' Lamp, by David B. Rushmore, Schenectady, N. Y.

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* Pamphlet copies distributed.

† Not approved by author in time for publication in this volume.

‡ Not furnished for publication.

The Institute Banquet.

The dinner at the Plaza, on Tuesday evening, Feb. 20, was a brilliant and enjoyable affair. The retiring President, Mr. Charles Kirchhoff, presided. Addresses were made by George Otis Smith, Director of the U. S. Geological Survey, who emphasized the cordial relations between that bureau and the Institute, and the mutual helpfulness of their work; Mr. W. L. Saunders, who, having just returned from a journey round the world, spoke interestingly of his observations in the industrial establishments of Europe; and Mr. H. Mortimer Lamb, Secretary of the Canadian Mining Institute, who gracefully expressed the congratulations of his Society. Col. Alexander M. Hay, a Canadian member of the Institute, recited amid laughter and applause a clever poetic burlesque, describing the discovery of the North Pole by a submarine vessel. Dr. Raymond, in responding to the toast of "The Ladies," claimed for the Institute the honor of having been the first national society to have ladies at its banquets, and quoted the following passage from a speech delivered by Alexander L. Holley at the first occasion of this kind, the Pittsburg banquet of 1879:

"Young man of the school, full of lore and anxious for hire, what is the vista of probabilities that fills your eye? Will you map out the metalliferous veins under the fair landscape from the rugged outcrops of the upturned rocks? Will you span the cañon, eroded throughout æons, with your gossamer steel bridge of yesterday? Will you compel a river which denudes a continent to build out its own ship-canal through its own delta with its own débris? Will you sever continents to make a highway for commerce? Will you coax out of ores with your deft alchemy the metals which the evolution of ages put in? Will you drive a train from the Orient to the Occident, following the sun and keeping company with the hours? Ah, my dear boy! these things you may do, but they are only means to an end. And that end you shall see down the long vista of the inevitable. There, with eloquent eyes and folded arms, sits a dear little woman! And, my boy, when those tender arms shall enfold you, and those eloquent eyes shall flash into your soul the potential caloric of a whole life,—*then* you will know what it is to be the lord of a fellow's wife!

"Two thousand years ago, the philosopher Cullimachus, wandering in the cemetery of Corinth, was arrested by a vision of prophetic beauty. It was only an acanthus plant, confined in a basket and covered with a tile, the struggling leaves curling through the meshes and wreathing themselves in graceful volutes under the covering stone. This was the decoration of a child's grave; but it was the prototype of the Corinthian capital. As out of a little grave grew the glory of decorative art, so out of ah! how many little graves, struggling through the meshes and repressed by the cold marble, perennially bloom the graces and the virtues of the higher life—that long-suffering, that patience, that elasticity, that sweetness, that association of the good and the beautiful, which is but another name for the fellows' wives!"

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PAPERS.

The Mineral Production and Resources of China.

BY THOMAS T. READ,* SAN FRANCISCO, CAL.

(San Francisco Meeting, October, 1911.)

I. Introduction.

When so much has been written upon a subject on which so little definite information is available as upon this, some reason must needs be assigned for adding to the volume of literature. A sufficient reason is found in the probable great future importance of China as a producer of mineral wealth. The present undeveloped state of mining in the Empire is due to many causes, among which the most important are the relatively simple needs of the population, the lack of transportation-facilities, the inelastic regulations governing the industry, and the superstitious reluctance of the people to make excavations which might disturb the spirits of the earth and air, or of ancestors.

Iron is the metal most in demand, yet the needs of the population, until recently, only made necessary a production of approximately 0.5 lb. of iron per capita per year. The present annual production of iron for domestic consumption, in the United States, is nearly 600 lb. per capita. The curious state of development of transportation in this interesting country has been a greater handicap upon the mining industry than upon any other. The point upon which it hinges is the absence of a road-system. Except for a few military roads, now almost impassable, there are no roads in China; that is to say, there is no land which is set aside as a right of way, belonging to the commonwealth. Throughout the agricultural districts, generally speaking, the entire area is in private ownership, and the lines of travel are between fields. There is a constant struggle for existence between the owners of the land and the traveling public, with the natural result that the so-called roads usually

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are simply lines along which it is merely possible to travel. Most merchandise is carried in baskets over the shoulder, pushed in wheelbarrows, or transported on pack-animals, to the nearest stream. Really cheap and efficient transportation is confined to the rivers, and great ingenuity is exhibited in utilizing them to the fullest extent. On the Lan river, for example, the boats, to use an expressive colloquialism, "can float in a heavy dew." The regions where mineral wealth is abundant are naturally but poorly supplied with navigable streams, and it is only where Nature has been so kind as to gather together in one place all the materials necessary for the extraction and reduction of the minerals that any considerable industry has been able to develop. Now railroads are penetrating all parts of the Empire, the streets built in the cities are being extended as roads through the surrounding territory, and the mineral industry is taking on a new aspect.

Theoretically, the mineral wealth of the Empire is the property of the central government, and is only worked by permission upon a royalty basis. Practically, this is difficult to enforce, and there have grown up relationships between the operators and the government which are complicated in the extreme. The development of the mining industry is correspondingly hampered, and it is to be hoped that a simple and direct code of mining-regulations may be put in force at no far distant date. The importance of the superstitions regarding graves and "Feng-Shui" has been over-emphasized. It has undoubtedly operated to restrict prospecting-work, but where valuable deposits are found it is always possible to have graves moved for a reasonable sum, and it is but seldom that a regard for "spirits" is allowed to operate to financial disadvantage. With the spread of education this factor will lose the limited importance it now possesses.

Easily first in the mineral wealth of China are coal and iron. Willis [54] (reference is to bibliography given at the end of this paper) has estimated that the anthracite-resources of Shansi and adjacent territory are equal to those of Pennsylvania, and while no estimate is possible of the total amount of bituminous coal, it is safe to say that it is also comparable with that of the United States. It is impossible as yet to estimate the iron-resources, for reasons given under the discussion of that metal,

but there is every reason to believe that they are extensive and valuable; some progress has already been made in their utilization on a modern scale. The Empire is the most important producer of antimony, and ranks high in the production of tin. The production of copper and zinc is already appreciable, and the production of petroleum, while as yet small, seems to have much of promise. The production of gold, silver, mercury, and other metals, while worthy of notice, seems to offer less hope of great increase by the introduction of improved methods of working. It will be evident from the following pages that the mineral production of China is at present of no little importance, and her known resources are great enough to offer ground for the belief that considerable development can be expected in the future.

During three years of residence in North China, I visited a large number of districts of which the mineral production is now considerable. It was not possible, of course, to visit all, and for the extreme south and southwestern parts of the Empire I have been dependent upon the statements of others, but especially upon the published reports of the notable group of French explorers and engineers who have extended their study of Indo-China to cover the adjacent areas. The report of Duclos,[13] and that of LeClere,[25] are especially important. Richthofen's monumental work [44] is of permanent value, as is that of the Carnegie Institution exploring party.[53] This latter deals only with general geology, but is of great interest as marking the first attempt to carry on topographic and geologic surveys in China with the precision attained in the United States. A very important paper is that of Willis,[54] who has made an admirable summary of the literature bearing upon the mineral resources of the Chinese Empire.

The bibliography which concludes this paper is not complete, nor has any effort been made to include all references to the subject. All the more important papers are included, but if any have escaped attention the correction will be welcome. Those who have carried on investigation in the Orient will appreciate the difficulty of the work. Important papers have been published in journals that are difficult or impossible to obtain, the native reports are entirely unreliable, and many areas of importance have never been adequately described. In

addition, there is much conflict between reports upon a single area, and often complete disagreement between figures as to present production. The following represents merely as close an approximation to the truth as now seems possible, and will require extensive revision in the light of future development. But having acquired, with much effort, an imperfect knowledge of the mineral resources of this great Empire, it appears a professional duty to place it in a form which may be of assistance to later workers in this broad field. Erroneous statements have probably been included, and it is hoped that the many engineers who have visited China will contribute further data.

In quoting from the reports of others the almost insurmountable obstacle of identification of place-names is constantly met. This is especially true of the French and German authors, whose spellings are often unrecognizable. Wherever possible I have given the latitude and longitude of the localities mentioned, except in the case of well-known places. Even this is unsatisfactory, for the most accessible accurate maps of southwestern China are French and the longitude is given in degrees east of Paris, which is approximately 2° 20′ east of Greenwich. The French notation has been followed in the case of points in Ssu-chuan, Yunnan, and Kweichou. This should be borne in mind in consulting the maps. In some cases where the spelling is obviously incorrect, or it has been impossible to identify the places, I have placed them within a semi-quotation mark 'thus,' and turned over the difficulty to the reader.

For convenience, reference is made by number in the text to the bibliography given at the end of the paper; [43] is a good geography of the Empire and [8] is a fairly complete bibliography of China.

II. Coal.

Coal is easily the first of the mineral resources of China. The great extent of the deposits has already been indicated and a conservative estimate of the present production is 15,000,000 tons annually. The casual visitor to North China, where the winter climate is rigorous, seeing the children of the villagers, armed with rake and basket, engaged in collecting every scrap of vegetable material that can be utilized as fuel, is likely to wonder why coal is not more generally used.

The reason is not recondite: the low cost of labor, the high cost of transportation, and the low scale of living put coal beyond the reach of the population in many regions. It is probably safe to estimate that one-half the cost of the food of an ordinary workman is chargeable to the fuel used in cooking it, and where the otherwise unemployed children can be sent out to gather grass and pull up the roots of the larger cereals, such as corn and *kao-liang*, there is little market for coal, except for industrial purposes. Near the mines, where coal is abundant and cheap, it is freely employed. The development of railroads, steamships, and industrial plants will not only create a greater market for coal, *per se*, but, by raising the scale of living through the higher wages paid for labor, will increase the consumption of coal for household purposes. The annual consumption of coal in the United States is approximately 3.5 tons per capita per annum; the consumption in China is approximately 1/25 ton per capita per annum. An approximate estimate of the present coal-production in China is given in Table I.

TABLE I.—*Approximate Estimate of the Present Coal-Production of China.*

Province.	Anthracite.	Bituminous.	Sub-Bituminous and Lignite.
	Tons.	Tons.	Tons.
Manchuria	.....	25,000	1,000,000
Chili	840,000	2,090,000	150,000
Shansi	4,000,000	25,000	
Shensi	.....	500,000	
Kansu	.....	500,000	
Shantung	300,000	500,000	
Honan	1,000,000	.....	
Ssu-chuan	.....	500,000	
Kweichou	.....	250,000	
Yunnan	.....	300,000	
Chekiang	.....	10,000	
Kiangsi	.....	700,000	
Hunan	.....	200,000	
Kuangtung	.....	50,000	
Kuangsi	.....	100,000	
Other provinces	.....	100,000	
Total	6,140,000 5,900,000 1,150,000	5,900,000	1,150,000
Grand total	13,190,000		

The sketch-map, Fig. 1, shows in general the coal-areas of China. This map, as well as Fig. 2, has some misspelled names, as it was impossible to arrange for revision of the proofs.

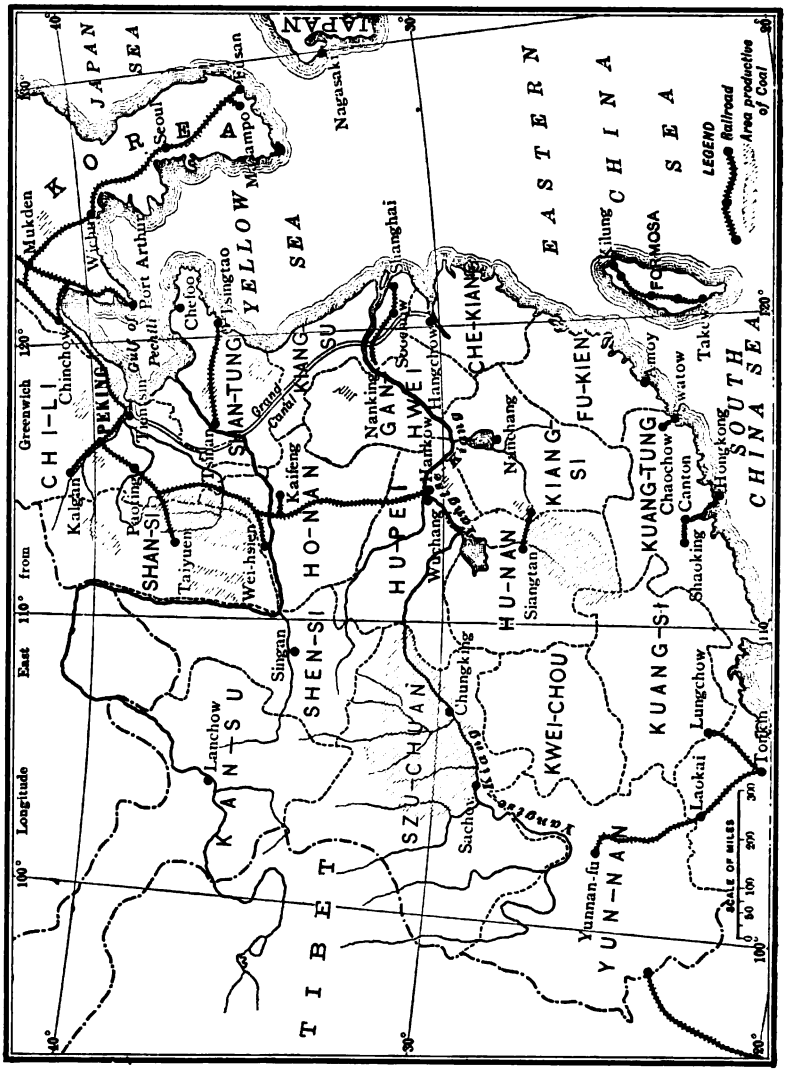

FIG. 1.—MAP OF CHINA, SHOWING COAL-AREAS.

The coals of China are as varied in quality as those of the United States, but this difference should be noted: the amount of lignite is comparatively small, and the proportion of anthracite to bituminous is relatively larger than in the United States. As previously noted, Willis has estimated that the anthracite-

resources of Shansi and the adjacent fields are practically equal to those of Pennsylvania. If this is in error it is probably upon the safe side, and the total coal-resources of the Chinese Empire seem likely, upon careful mapping, to compare favorably with those of the United States.

In the space at my command it would not be possible to give

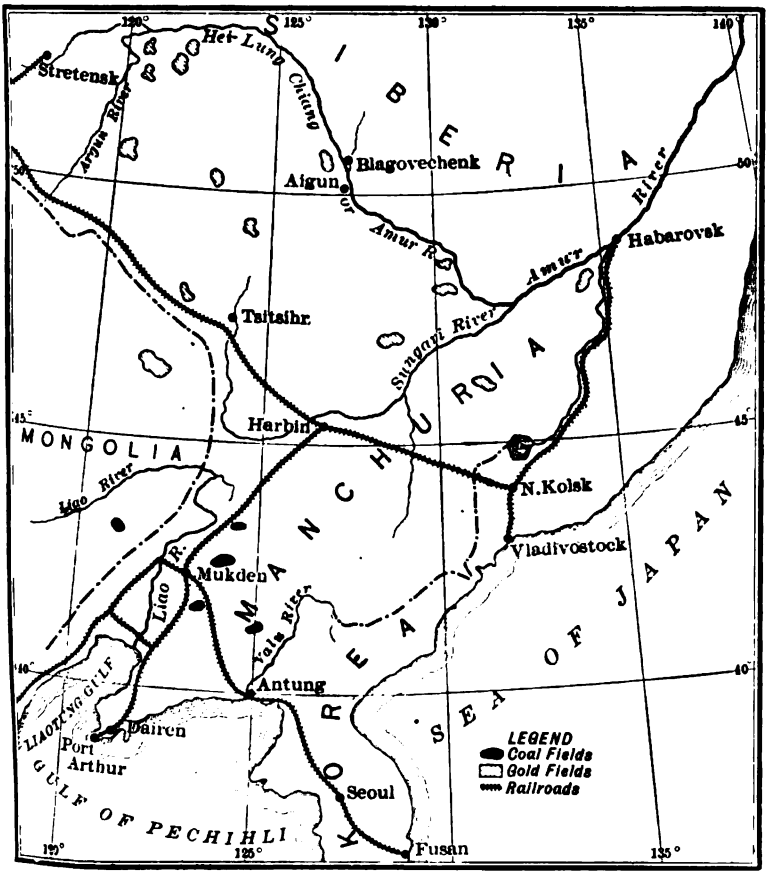

FIG. 2.—COAL-FIELDS AND GOLD-FIELDS OF MANCHURIA. (AFTER PURINGTON.)

more than a brief outline of each of the important fields. The number of fields is so great that an attempt at classification would lead to too great complexities, and it will be simplest to consider them briefly by provinces.

In Manchuria one large mine is now in operation at Fushun. This field, seen just above the word "Mukden" on the sketch-

map, Fig. 2, has been described in detail, [38], [30], so nothing further will be given here than to state that the coal is a sub-bituminous of excellent quality. The mines, owned and operated by the South Manchuria railway, had a production in 1910 of 830,328 tons, and are expected to reach 1,000,000 tons per year when the second of the two pairs of deep shafts (18 and 20 ft. in diameter) are in full working-order. More recently the mines at Pen-hsi-hu, on the Antung-Mukden railway, have been developed; but, although I passed through the town in the autumn of 1910, I have been unable to obtain figures as to the production, which is probably small as yet, but is likely to develop when the standard-gauge line between Mukden and Antung is in operation. The chief engineer at Fushun stated in 1909 that the Pen-hsi-hu coal is a friable semi-bituminous, occurring in Jurassic strata, and not of especially good grade, but more extensive development may have disclosed seams of better quality. The same remarks will apply to Sai-ma-chi (125° E., 41° 30′ N.); owing to its distance from the railroad no very serious attempts have been made to develop these mines. All mines in the South Manchuria railway zone are to be developed jointly by Japanese and Chinese.

Just east of Kwan-cheng-tze (125° E., 44° N.), coal similar to that at Fushun occurs in several places, and though the attempts at working have not been very successful as yet, the seams at this place are likely to become of great importance, as the branch railroad to Kirin cuts directly across them. West of the Liao river (somewhere about 122° E., 43° N.) a valuable and important field is said to exist, but I have not visited the locality and have no definite knowledge of it. Some time since the Imperial Railways of North China attempted to exploit some seams of a true lignite a short distance north of the Great Wall, but they proved to be of too poor quality. Other localities where coal occurs are Yentai and Wu-hu-tsui, but the production is unimportant. Thin seams of coal can be seen in the cuts along the trans-Siberian railroad, and the Bureau of Mines for Manchuria has published a long list of places where coal occurs in Manchuria. The coals of the southern part of the province were described by K. Inouye in 1905.[30a] Coal occurs widely throughout the area; that at Fushun and near Kwan-cheng-tze is sub-bituminous and of Tertiary

age; in the other districts the coal is semi-bituminous and of Jurassic age. No closer classification can be attempted as yet. The native consumption is generally supplied by small local mines. The South Manchuria railway and a considerable portion of the shipping-trade of Dalny (Tairen) are supplied by the Fushun mines. As the workable coal in the Fushun field has been estimated at 800,000,000 tons, Manchuria is well supplied with coal. But the Manchurian coals are very friable, furnishing but a small proportion of lump-size, and no good coking-coal has yet been found. The composition of the coals of Manchuria is given in Table II.

TABLE II.—*Analyses of Manchurian Coals.*

No.	Locality.	Moisture.	Volatile Hydrocarbon.	Fixed Carbon.	Ash.	Sulphur.	Analyst.	Remarks.
		Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.		
1	Fushun.....	6.30	39.34	52.90	3.18	0.27		Average of 7 published analyses.
2	Fushun.....	4.43	40.33	48.89	6.35	1.00	C. H. Wang	
3	Wu-hu-tsui...	2.70	11.42	76.69	9.19	0.50	Geological Survey of Japan.	
4	Pen-hsi-hu...	0.96	21.66	66.06	11.32	0.84		Average of 3 published analyses.
5	Yentai.....	1.07	14.22	74.98	9.75	0.66		Average of 3 published analyses.
6	Sai-ma-chi....	1.39	25.58	60.47	12.25	0.80		Average of 3 published analyses.

Chili, which immediately adjoins Manchuria on the south-west and is the metropolitan province, is now the most important producer of coal, as can be seen from Table III., which gives a summary of the coal-output of Chili province for 1909 by K. Y. Kwong, chief mining engineer for the Chili Province Bureau of Mines, Tientsin. The composition of some Chili coals is give in Table IV.

TABLE III.—*Production of Coal in Chili Province for* 1909.

Lignite.

		Tons.
Jehol district (118° E., 41° N.) coal-fields.....	Yen-pao-shan, Wo-chia-tze, Shih-t'ou-fen,	150,000
Total lignite.....		150,000

Anthracite Coal.

Kin-Han railway district coal-fields	Tai-an, Fangshan-hsien (116° E., 39° 45′ N.), Sha-ho-hsien (114° 40′ E., 37° N.), Lin-ming-kwan, Han-tan-hsien,	600,000
Peking-Shanhaikwan railway district coal-field,	Shi-men-tsai,	50,000
Peking-Kalgan railway district coal-fields	Sin-pao-an,	20,000
	Peking Western Hills (115° 45′ E., 40° 15′ N.),	150,000
Total anthracite		840,000

Bituminous Coal.

Peking-Shanhaikwan railway district coal-fields	Kaiping C. E. & M. Co., Ltd., mines, Lanchow mines,	1,400,000
Kin-Han railway district coal-fields	Lingshan,	50,000
	Ching-hsing (114° E., 38° N.),	160,000
	Lincheng (114° 30′ E., 37° 30′ N.),	100,000
	Tzê-chow (114° 20′ E., 36° 30′ N.),	100,000
Peking-Kalgan railway district coal-fields	Hsuen-hua-fu (115° 10′ E., 40° 50′ N.),	40,000
Lan-Ho and Jehol districts coal-fields	.....	100,000
Chow-yang-fu coal-fields		140,000
Total bituminous coal		2,090,000

TABLE IV.—*Analyses of Some Chili Coals.*

No.	Locality.	Moisture.	Ash.	Fixed Carbon.	Volatile Hydrocarbon.	Sulphur.	Fixed Carbon Ratio.	Analyst.	Remarks.
		Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.			
1	Tongshan, C.E. & M. Co.	1.00	16.67	56.78	25.55	1.01	2 22	C. H. Wang.	Coking, bituminous.
2	Tongshan, C.E. & M. Co.	0.84	18.02	57.19	23.95	1.46	2.38	C. H. Wang.	Bituminous.
3	Tongshan, C.E. & M. Co.	1.37	21.72	53.81	23.10	2.55	2.33	C. H. Wang.	Non-coking, bituminous.
4	Tongshan, C.E. & M. Co.	0.93	12.29	59.75	27.03	3.90	2.21	C. H. Wang.	Coking, bituminous.
5	Tongshan, C.E. & M. Co.	0.90	13.55	70.33	15.22	2.26	4.62	C. H. Wang.	Coking, bituminous.
6	Tongshan, C.E. & M. Co.	0.77	18.59	53.28	27.40	1.11	.....	Average company	analysis furnished by
7	Linsi, C. E. & M. Co.	0.77	19.18	51.97	28.05	0.88	.....	Average company	analysis furnished by
8	Ching-Hsing	0.54	13.49	61.16	24.81	2.26	2.46	C. H. Wang.	Coking, bituminous.
9	Chai-t'ang	2.11	6.67	68.23	22.99	0.19	2.97	C. H. Wang.	Non-coking, bituminous.
10	114° 30′ E., 38° 45′ N.	1.20	12.97	80.50	5.43	0.97	14.81	C. H. Wang.	Hard, dry anthracite.

The lignite and bituminous coal of the Jehol district is produced by native methods, the nature of which can be inferred from the photographic views, Figs. 3 and 4, taken in the

Western Hills. The Kin-Han (Peking-Hankow) railway district anthracite-field and the Peking-Kalgan railway district field are northern and southern portions of one field lying a short distance west of Peking. Hoover [17] and Woo [56] have described the Chinese Engineering & Mining Co. mines at Tong-shan and Linsi in detail. These are the largest and most important coal-mines in China. They are owned by an English company, but it has been proposed by the gentry of the province that the concession be purchased by the Provincial government. Since the descriptions were written, the mines have been much developed and improved. They produce nearly all of the total of the 1,400,000 tons estimated above, as the semi-official Lanchow mines, in the same field, have only recently been started and, though well equipped, seem unable to operate at a profit. The net profit of the Chinese Engineering & Mining Co. mines for the year ending February, 1910, is given as £243,300. The Ching-hsing mines, on the railroad from Shih-chia-chuang to Tai-yuan-fu, are worked under German supervision, and have both Chinese and German capital. The production in 1910 was 150,000 tons. The Lincheng mines are operated to supply the Kin-Han railway with fuel, and are under the supervision of K. Y. Kwong. They have an output of 800 tons per day. The coal-field at Tzê-chou has been described by Drake.[10] The mines at Hsuen-hua-fu have only recently been opened to supply the Peking-Kalgan railway with fuel, and their production will probably increase considerably during the next few years; there is already a considerable production by native methods in this and adjacent districts. It is obvious that it would be possible to devote much space to a description of the mines of this one province, but they are so accessible and comparatively well known that, in spite of their great importance, I shall not discuss them further. The analyses in Table IV. show the characteristics of these coals, and it will at once be noticed that they grade by degrees from bituminous into anthracite.

In Shantung coal occurs in many places, but the larger part of the production comes from the mines owned by the Shantung Bergbau Gesellschaft, at Po-shan (118° 0′ E., 36° 45′ N.) and Fang-tze; 252,816 tons of anthracite coal having been produced at the former place in 1910, and 230,064 tons of

bituminous at the latter. These mines have washing-plants. Some difficulty has been found in working, owing to the faulting and disturbance of the beds, and the native papers say that there is little profit in their operation. In this, as in every other coal-field in China, there is a large amount of native mining upon a small scale. Farther to the southeast, near Yi-hsien (118° 36′ E., 35° 0′ N.), is a bituminous field which is said to be larger and better, but which I have not visited. The production of the native mines is already important. A large native company, called the Chung Hsing Kung Ssu, has been formed, machinery procured from Germany, and a railroad constructed from the mines 35 miles to the Grand Canal. This will probably be extended to connect with the Tientsin-Pukou railroad, and when the latter road is in operation the production of coal in this district should become important.

Much has been written concerning the coal-fields of Shansi, and having spent some time in visiting the more important localities, the temptation is strong to describe them fully. But since others, [12], [54], [44], [45], have discussed them in more or less detail, I shall only refer to them briefly. These anthracite seams are the most striking coal-beds in the Empire, as they are so thick, so little disturbed, so well exposed, and so widely distributed, having an extent of nearly 200 miles north and south, and from 25 to 30 miles east and west. There are several seams, one of which is especially thick and persistent. Richthofen [44] estimated the area of the field as 13,500 sq. miles, and Drake [12] estimated the average workable thickness of the seams as 22 ft. As the beds are frequently but slightly inclined, this corresponds to a yield of over 22,000,000 tons per square mile of workable area, so it is safe to estimate that the anthracite-resources of this part of China are at least equal to those of the United States. Mining in this field is under the control of the Pao-Chin Mining Co., which was formed by the gentry of the province to repurchase the concession of the Peking Syndicate. Several shafts are making a small production, most of the present output coming from native workings. The area controlled by the Pao-Chin Mining Co. is of immense importance, and it is to be regretted that so little progress is being made. There were, in 1910, no trained engineers in the employ of the company, though there are numbers

of properly qualified Chinese engineers available. The analyses given in Table V., most of which were made by C. H. Wang, exhibit the character of the coals. An interesting feature is the high content of phosphorus, and the importance which this has had in the native metallurgy of iron is discussed in the section devoted to that metal. Shansi also possesses considerable resources in bituminous and semi-anthracite coal, which is produced both to the east and west of Tai-yuan-fu and sold in considerable quantity in that city. Analyses are given in Table V.

TABLE V.—*Analyses of Shansi Coals.*

No.	Locality.	Moisture.	Volatile Hydrocarbon.	Fixed Carbon.	Ash.	Sulphur.	Phosphorus.	Analyst.	Sp. Gr.	Remarks.
		Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.			
1	Chuang-chuang-kou,	0.91	8.20	78.75	12.14	0.67	0.476	C.H.Wang.	.....	Semi-anthracite, does not coke.
2	Hon-ho-kou.....	0.94	6.15	85.70	7.21	0.59	n. d.	C.H.Wang.	.....	Hard dry anthracite.
3	Han-ho-kou.....	0.38	8.55	84.00	6.72	0.409	0.23	F. N. Lu.	1.38	
4	Tuan-chia-kou.....	0.76	6.44	79.10	13.66	0.683	1.80	F. N. Lu.	1.35-1.38	
5	Lao-hsien-chêng-kou,	0.66	4.49	89.50	5.46	0.509	2.53	F. N. Lu.	1.4	
6	Nan-t'ien-mêng.....	2.01	7.05	81.35	9.61	0.867	n. d.	F. N. Lu.	1.3-1.5	
7	Meng-tse.n-chêng.....	0.33	4.39	89.25	6.03	0.405	2.15	F. N. Lu.	1.3 high	
8	Chuang-chuang-kou,	14.75	19.28	52.42	13.45	0.890	n. d.	F. N. Lu.		Coked in crucible, high moisture probably due to oxidation.
9	Chang-tsai-kou.....	0.50	14.20	75.89	9.41	0.258	n. d.	C.H.Wang.	.....	Semi-bituminous, coked in crucible.
10	Tung-chia-chuang.....	1.73	10.72	84.22	3 33	1.150	n. d.	C.H.Wang.	.....	Semi-anthracite, does not coke.
11	.....	1.93	3.45	81.44	14.17	0.35	n. d.	.....	.....	Average of 6 analyses by Drake.[12]
12	.....	2.91	86.80		9.88	0.41	n. d.	.....	.....	Average of 6 analyses by Shockley.[47]

Samples 1 to 8 were taken throughout the district (113° 30′ E., 38° N.); sample 9 from (112° 40′ E., 37° 55′ N.); and sample 10 from (112° 40′ E., 37° 50′ N.).

In Shensi, which adjoins Shansi on the west, extensive coal-fields are known to exist, and Richthofen, [44], [45], thought that the bituminous-fields to the west of Tai-yuan-fu were of equal extent and importance as the anthracite-fields to the east. This is possibly too optimistic, but they are certainly very great. I have estimated the production of this province as 500,000 tons, but this is problematical, as the area is so little known. The same remarks apply to Kansu, which adjoins Shensi on the west. I have, for this reason, omitted Mongolia from Table I., on page 7, though it is known to possess coal-seams in

those portions which adjoin Chili and Shansi. Passing directly south, to Ssu-chuan, Kweichou, and Yunnan, our knowledge is in a similarly unsatisfactory state. Richthofen,[45] who only traversed the northeastern part of Ssu-chuan, says, in substance, that coal is very generally worked throughout the province, as the Mesozoic strata are extensively folded and are cut across by the rivers, thus conveniently exposing the seams. So far as I can learn, the coal here is not of as good quality as other deposits more favorably situated with respect to the larger markets, which, together with the difficulties of transportation on the Yangtze, restricts production to the amount required for local needs. A few years ago, a British company secured a concession at Wan-hsien (108° 30′ E., 31° 0′ N.) and installed modern machinery, but it has now been handed over to a native company and is worked by native methods. Baber,[1] Hosie,[18] Duclos,[13] LeClere,[25] and others have noted many places where coal is worked in Ssu-chuan. Duclos has discussed at some length the occurrence and methods of working coal in Ssu-chuan; his remarks are best summarized by a quotation from his report: "Le charbon se trouve presque partout dans le province, donnant lieu à de petites exploitations qui subviennent aux besoins locaux." LeClere,[25] who has a favorable opinion of the coal-resources of southwestern China, says that coal occurs at four horizons, from lower Palæozoic to Rhétic, and is widely distributed over a quadrilateral area bounded by 'Lao-Kay' (101° 31′ E., 22° 30′ N.), 'Yunnan-hsien' (100° 30′ E., 23° 0′ N.), 'Tchao-toung' (101° 31′ E., 27° 30′ N.), and 'Kouei-Yang-hsien' (104° 20′ E., 26° 20′ N.). He thought that the field most favorable for exploration is that lying to the west of Mengtze (101° 0′ E., 22° 45′ N.). Duclos [13] mentions four places in Kweichou where coal is produced; 'Mao-py', 'Tchen-lin' (103° 15′ E., 26° 0′ N.), 'Choui-tang-pou', and 'Ma-lou-kio' (102° 27′ E., 26° 58′ N.).

The anthracite-field of Shansi extends southward into Honan province and at Ching-hua-hsien, 113° 40′ E., 35° 15′ N. approximately, the Peking Syndicate has several shafts, equipped with modern machinery, in operation. This company has had a good many vicissitudes, but is now meeting with success; the production during 1910 is given as 357,205 tons. There is a good deal of production by native methods in this province.

The great southern coal-field lies to the east of the Hsiang river, in Hunan and Kiangsi provinces. The greater part of the field is in the former province, but the most important producer, the collieries of the Han-Yeh-P'ing Iron & Coal Co., are at P'ing-hsiang (113° 50′ E., 27° 30′ N.), in Kiangsi. A view of the P'ing-hsiang colliery, showing two banks of coke-ovens and the washing-plant, is given in Fig. 5, and a nearer view of one bank of ovens is shown in Fig. 6. The steel head-frame of the shaft is shown in Fig. 7. This coal is a bituminous coking variety (with associated thin seams of anthracite), which contains 28 per cent. of ash as mined, but after washing and drying an average analysis furnished by the company gave:

	Per Cent.
Volatile hydrocarbons,	22.35
Fixed carbon,	68.90
Ash,	8.70
Sulphur,	0.10

It yields excellent coke, which supplies the blast-furnaces at Han-Yang and the general market; more than 107,000 tons having been produced in 1909. Further details can be found by consulting [37] and [46]. The production for 1910 is given as 610,000 tons. The coal-fields extend west and south from this point for a great distance, and Richthofen [45] says that southward the coal is anthracite and of better quality. Transportation is difficult, owing to the shallowness of the rivers; so development has lagged; but when the Canton-Hankow railway, now under construction, is in operation, this field, which I regard as only second in importance to the Shansi field, is likely to develop greatly. Some of the most important mineral regions in China lie to the west of the projected railway-line, and the transportation-facilities thus afforded should lead to a great increase in mineral production.

The coal-fields of Chekiang are of little importance. In Kwangtung and Kwangsi coal is mined at a number of places, but I have no personal knowledge of this area. My impression that these coal-deposits are worked because of their proximity to important centers of trade, rather than because of their superior quality, is not improbably correct.

Despite the importance of the coal-fields of China, it is impossible to afford more space to their discussion. Reference should

be made to the papers quoted for further details. In conclusion, it should be said that the coal-fields of China are of great extent, the coal is generally of good quality and the fields are widely scattered, so that no parts of the Empire are far distant from the sources of supply. In extent and quality the coal-resources of the Empire compare favorably with those of the United States. As a rough comparison, it may be said that Chinese coals are slightly younger than those of North America, most of the fields being upper Carboniferous or Permian. In the north, Jurassic and Tertiary coals occur, but except for the Fushun field, are of little importance as yet. There has been a good deal of controversy over the exact age of these coals, but it is of little interest to mining engineers. Bituminous coking-coal is very common; coke made by native methods can be obtained almost anywhere in the Empire. When made from washed coal, the resulting coke is of excellent quality, and will afford an abundant supply for the smelting industries which are likely to develop. The anthracite is of excellent quality, but the bituminous is often friable, yielding an excessive proportion of fine coal. When worked on a large scale this can be washed and converted into coke. The Chinese custom is to make the dust into briquettes with clay as a binder, which are dried and burned. In recent years the number of mines equipped with modern machinery has become comparatively great, and the present supply amply meets the demand. The mines of the Fushun Co. and the Chinese Engineering & Mining Co. now chiefly supply the railway- and shipping-trade of North China, because of the superior transportation-facilities which they enjoy. These two companies also send coal to the cities along the SE. coast, competing with Japanese coal. Coal-production throughout China exhibits a healthy and vigorous growth. The annual production of the Empire, as shown in Table I., is estimated at 15,000,000 tons; but this amount is only approximate. The amounts given for Yunnan and Ssu-chuan, for example, should be designated as guesses rather than estimates, and the figures occasionally seen in statistical tables giving the production of the Empire to the nearest thousand tons are totally misleading in the false appearance of accuracy which they present.

FIG. 3.—MOUTH OF A COAL-MINE WORKED BY NATIVE METHODS. THE SMALL BOY ILLUSTRATES THE SIZE OF THE OPENINGS.

FIG. 4.—BASKETS USED IN TRANSPORTING COAL FROM WORKING-FACES TO PIT-MOUTH IN MINES WORKED BY NATIVE METHODS.

FIG. 5.—P'ING-HSIANG COLLIERY, KIANGSI.
TWO BANKS OF COKE-OVENS IN FOREGROUND, WASHING-PLANT TO RIGHT.

FIG. 6.—COKE-OVENS, P'ING-HSIANG COLLIERY, KIANGSI.

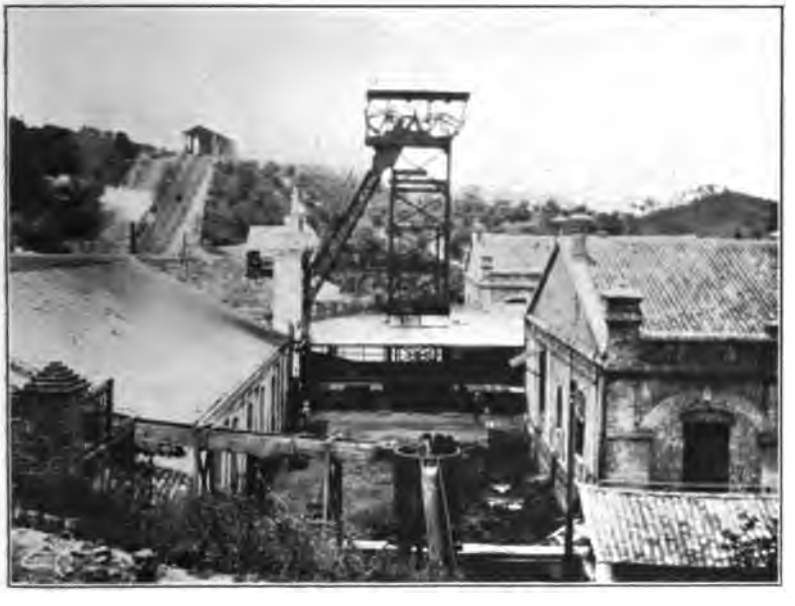

FIG. 7.—HEAD-FRAME, P'ING-HSIANG COLLIERY.

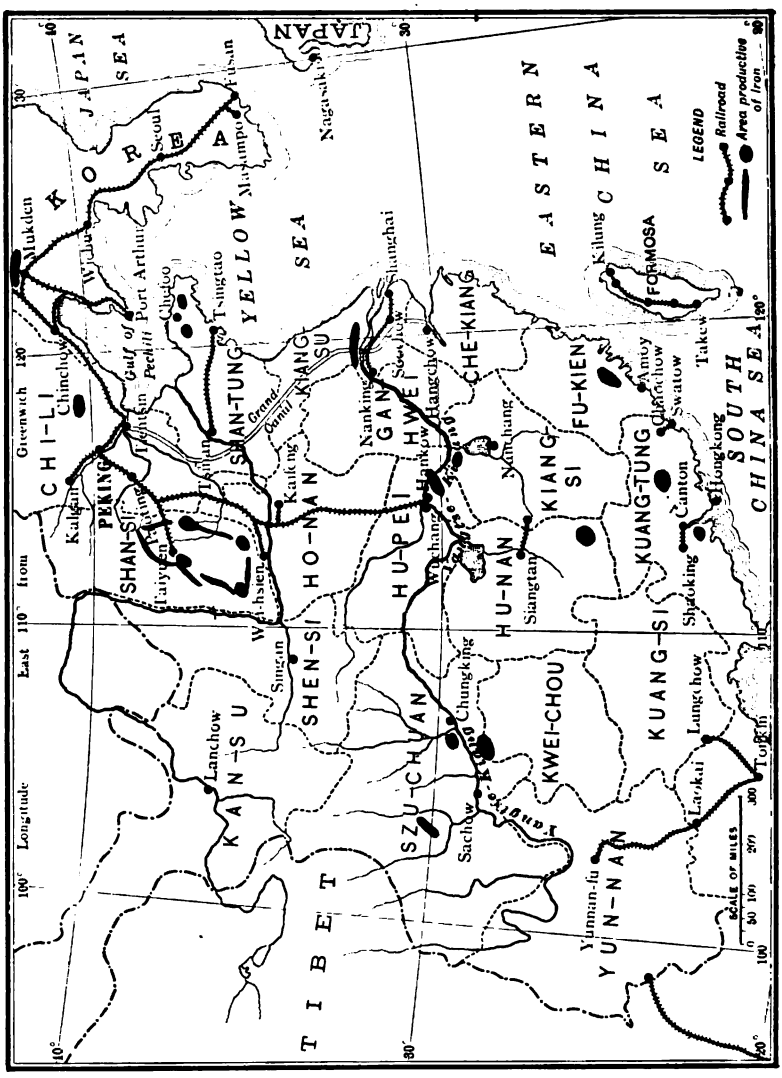

FIG. 8.—MAP OF CHINA, SHOWING IRON-ORE CENTERS.

FIG. 9.—HOISTING IRON-ORE FROM A SHALLOW CIRCULAR SHAFT, NORTH OF P'ING-T'ING CHOU, SHANSI.

FIG. 10.—A BATTERY OF CRUCIBLES REDUCING IRON-ORE, SHANSI. WHEN THE FRONT HAS BEEN CLOSED IT WILL BE READY TO LIGHT.

FIG. 11.—FURNACE FOR REMELTING THE PIG-IRON FOR CASTING, SHANSI. THE UNUSED CRUCIBLES ARE ON THE BOTTOM AND PILED AT THE BACK, USED CRUCIBLES ARE SEEN AT THE SIDE; THE BLOWING-APPARATUS IS AT THE BACK.

FIG. 12.—TRANSPORTING *Kuo* (CAST-IRON PANS) TO MARKET. CLAY FOR CRUCIBLES IS PLACED IN THE ROAD TO BE PULVERIZED BY THE MULES. THE HEAPS ON EACH SIDE ARE WASTE FROM THE SMELTERS.

III. Iron.

Iron is the second in importance of all the mineral resources of China, and the security of the future of China as a mineral-producing nation is easily appreciated, since it is founded upon an abundant supply of coal and iron, the two bases of industrial development. In the case of other minerals it may be inferred, without falling into serious error, that practically every occurrence of importance has been worked, to a slight extent at least, by native methods. But this is not true of iron-ore deposits. The great importance of the lack of transportation-facilities upon the development of mineral wealth has already been mentioned, and it is in the case of iron that its influence has been most marked. In the native method of iron-smelting such large quantities of coal are used that it is necessary to have both coal and iron-ore in close proximity in order to permit the development of a smelting industry. This condition obtains in several provinces, but most notably in Shansi, where it has been described by Richthofen, [44], [45], and Shockley.[47] But while these deposits are adapted to the native methods of working, they are not at all suited to modern methods, where a large supply of iron-ore of uniform quality and high iron-content is necessary. It is reasonably certain that such deposits exist in numerous places; but, except in the case of the Ta-yeh mine, to be described later, they have not yet been developed because the necessary supply of coal does not exist near-by. A sketch-map showing the iron-centers in China is given in Fig. 8. A description of the deposit at Ta-yeh will be given, followed by a list of the principal localities throughout the Empire where iron-ore is known to occur. I will preface this by a description of the P'ing-t'ing-chou district in Shansi, the principal one in which iron is produced by native methods. The accounts of this by Richthofen have already been mentioned; what is here given, however, is based upon notes made during the winter of 1910. The analyses given of the raw materials and products have been made by my former students, C. F. and C. H. Wang and F. N. Lu.

The iron-ores of Shansi are limonite and hematite, occurring in shales and sandstones of Carboniferous age; the varieties of method of occurrence are so numerous that to attempt their

description would require too much space. Usually they are in masses of no great size, commonly in or near a disturbed zone in the strata, or else in beds or flat veins, from a few inches to not more than 3 ft. thick, of limited extent. It follows, therefore, that no sufficient supply of uniform enough quality can be obtained from the Shansi deposits, so far as yet explored, to form the basis of blast-furnace work on a large scale. More recently it has been reported that on the southern border of this district, in Honan province, a suitable deposit has been found, and it has been proposed to erect there another government iron-works, similar to the one described later. Two analyses of the ore from T'ai-yang, near Tze-chou-fu, as given by Shockley,[47] are shown in Table VI.

TABLE VI.—*Analyses of Iron-Ore from T'ai-Yang.*

	I.	II.
Fe	53.88	45.50
SiO_2	4.67	11.15
Al_2O_3	3.46	6.42
MnO_2	0.57	0.51
CaO	2.21	5.50
MgO	trace	0.25
P_2O_5	0.57	1.08
S	0.074	0.016
CO_2	9.37	2.70
H_2O	2.20	7.35
Analysts,	Edward Riley.	Pattinson and Stead.

The ore, mined through shallow round or rectangular shafts, Fig. 9, is broken into small pieces and hand-sorted into several grades, which are sold to the smelting-plants. Here it is mixed with 50 per cent. of its volume of coal and packed into cylindrical crucibles, 5 in. in inner diameter, and usually 45 in. high, Fig. 10. From 250 to 275 of these crucibles are set upright in a rectangular furnace, about 12 by 6 by 4 ft. Air-space is secured at the bottom by a layer of broken crucibles, over which is placed a layer of coal; then the crucibles are set in place, with coal between them; the front side is closed, the whole is covered over with coal and allowed to burn by natural draft for three days. The crucibles are then removed and the contents taken out. This operation usually involves breaking the bottom part of the crucible, which now contains an irregular "bloom" of iron of very variable composition (Table VII.),

irregular fragments of iron, earthy residues, and a certain amount of coke. The bloom is sold to the makers of wrought-iron, the small pieces of iron are sold to the makers of cast-iron, and the coke is used in the manufacture of crucibles. It should be noted that the product of this method of melting is not pig-iron, in the ordinary sense of the word, as it contains very little carbon, and is malleable. The bloom is worked into wrought-iron by heating in a wood fire and hammering until it is worked down into a rectangular ingot, which is then sold, and either manufactured locally into various objects and utensils, or shipped in the ingot form to all parts of the Empire. The small pieces of iron are mixed with coal and placed in crucibles, about 7 by 14 in., and from 50 to 80 of these are placed in a smaller furnace, Fig. 11, blown by hand. When the iron is melted the covering of the furnace is removed, the crucibles are taken out, the contents of several crucibles are poured into one, and this is then poured into molds, which have previously been prepared with extraordinary skill. In this way various cooking-utensils, especially *kuo*, are cast, often of remarkable thinness, as the castings contain as much as from 5 to 7 per cent. of phosphorus, which has been taken up from the coal during the reduction and remelting.

TABLE VII.—*Average Phosphorus- and Sulphur-Content of Native Pig-Iron of P'ing-t'ing-chou, Shansi, China.*

Sample No.	Locality.	Phosphorus.	Sulphur.	Remarks on Samples.
		Per Cent.	Per Cent.	
1	Li-chia-chuang.....	1.7314	0.5489	Clean, malleable.
2	T'ao p'o.....	0.4671	0.178	Dirty, malleable.
3	Yang-chia-chuang..	4.8400	0.4577	Rather clean, brittle.
4	Nan-yao-kou.....	2.6066	0.2486	Dirty, globular.
5	Chien-mu-p'ing.....	0.8527	0.1254	Dirty, porous, brittle.
6	San-ch'uan.....	0.6854	0.1365	Clean, porous, brittle.
7	Yang-shu-kou.....	1.5075	0 6140	Dirty, porous, slightly malleable.
8	San-tu.....	0.4645	0.4945	Dirty, porous, slightly malleable.
9	Ing-Ying.....	3.5700	0 6375	Clean, hard, massive.
10	Han-ho-kou.....	3.7645	0.2999	Clean, hard.
11	Wu-tu.....	0.7315	0.2155	Clean, malleable.
12	Yang-chü Hsien ...	6.9540	0.2196	From a cast-iron bar.
	Average.....	2.3479	0.3480	

Analyses by Cheng-Fu Wang.

Note in Table VII., sample No. 12, the high percentage of phosphorus in the cast-iron bar. It is not at all improbable

that the high phosphorus-content of the Shansi coals, Table V., samples Nos. 5 and 7, has been the chief factor in the great development of the industry, as it has afforded an easy means of securing the high-phosphorus iron necessary in making thin castings. Fig. 12 shows the method of transporting cast-iron pans, *kuo*, to market; the heaps on each side of the road are waste products from the smelter; and the road between is filled with lumps of clay. Formerly the iron-products of this locality found their way through the channels of trade to nearly all parts of the Empire. Now the trade in native iron exists chiefly in the interior, as foreign scrap-iron and steel, especially old horse-shoes, are imported in large quantities into all the treaty-ports; and because of their superior quality and low price tend to drive out the native product. The native industry, therefore, is steadily dwindling, though it will continue to exist until foreign cooking-utensils displace the high-phosphorus cast-iron *kuo*. Shockley [47] has called attention to the amount of decrease between the time of Richthofen's visit and his own. Table VIII. gives the production, as estimated by Shockley, of the chief districts of Shansi in 1901.

TABLE VIII.—*Annual Production of Native Iron in the Chief Districts of Shansi.*

	Tons.
Yu-hsien,	4,500
P'ing-ting-chou,	18,000
Yiu-ch'eng,	6,000
Kao-p'ing-hsien,	4,000
Tse-chou-fou,	13,333
Yang-ch'eng,	2,000
Ch'in-shui,	1,415
Tai-yuan-fu,	2,000
Total,	51,248

The following notes of the only modern steel-works existing in China at present were made during a visit to its iron-mines and metallurgical plant during the summer of 1908.

The blast-furnace and steel-plant of the Han-Yeh-P'ing Iron & Coal Co. is situated at Hanyang, just across the Han river from Hankow, in Central China. Its iron-ores are mined at Ta-yeh, about 50 miles SW., and its coke is obtained from its colliery of P'ing-hsiang, in Kiangsi province, more than 300

miles distant. The relative positions of the properties are shown on the sketch-map, Fig. 13.

The iron-ores at Ta-yeh occur about 15 miles west of the Yangtze river, and lie along the contact between a marble and an intrusive body of a dark gray syenitic rock, Fig. 14. There is no direct evidence in the neighborhood as to the age of the

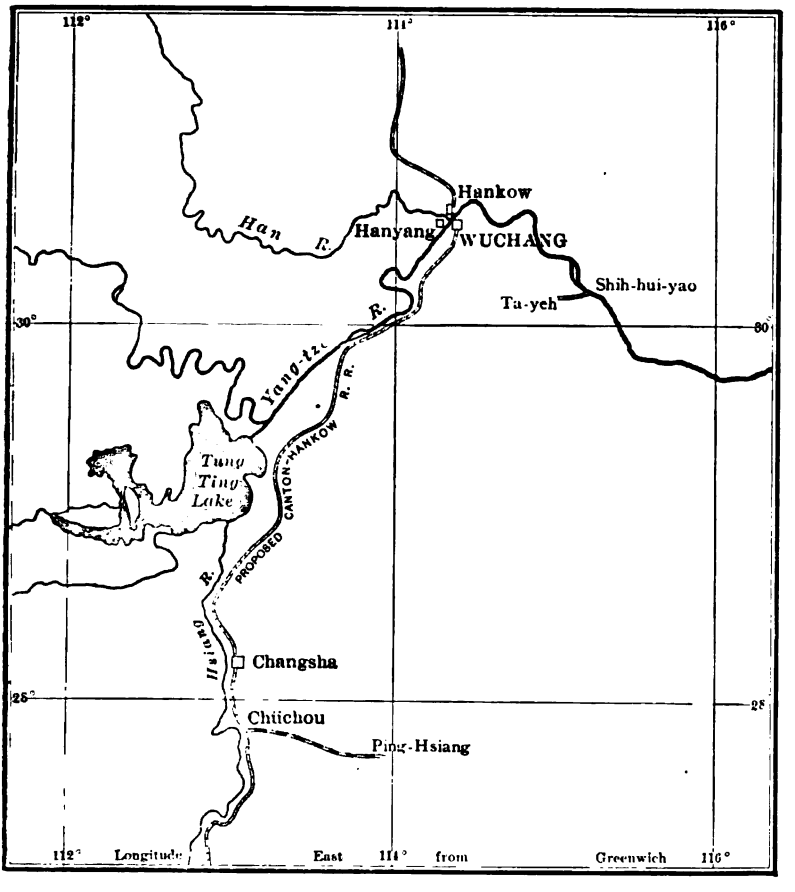

FIG. 13.—SKETCH-MAP SHOWING THE RELATIVE POSITIONS OF THE PROPERTIES OF THE HAN-YEH-P'ING IRON & COAL CO.

marble, but on lithological grounds it seems identical with the limestone occurring near Nanking, which Richthofen [45] has called the 'Sihia.' (He has translated the Chinese characters incorrectly; it is properly 'Hsi-hsieh.') Richthofen has suggested the age as Devonian, but the closely-associated coal in

both localities makes it much more probable that it is rather to be regarded as Carboniferous, since the Devonian is generally very poorly developed in China. There is much doubt as to exact correlation of China fossil fauna with that of other countries, and in nearly all cases coals in China belong to the Carboniferous, or the immediately overlying series. If these two are identical, it would then appear that this limestone is a horizon for iron-ores in the Yangtze valley, as they are associated with it in notable amount near Nanking, as will be noticed later.

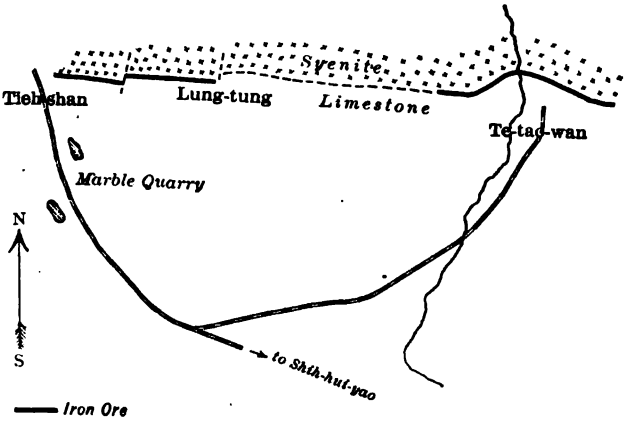

FIG. 14.—IRON-ORE DEPOSIT AT TA-YEH.

The ore is a good quality of hematite of about the following range of composition, furnished by the Han-Yeh-P'ing Iron & Coal Co.:

Iron Ore, Ta-yeh.

	Per Cent.
Fe	60 to 62
P	0.05 to 0.25
S	0.05 to 0.12
SiO_2	3 to 5
Al_2O_3	1 to 2
Mn	0.2 to 0.4
Cu	0.05 to 0.25

At one place it is slightly magnetic, apparently having been partly reduced to the magnetic oxide by the action of reducing solutions, which have deposited small amounts of copper and iron sulphides along the foot-wall. During the Ming dynasty,

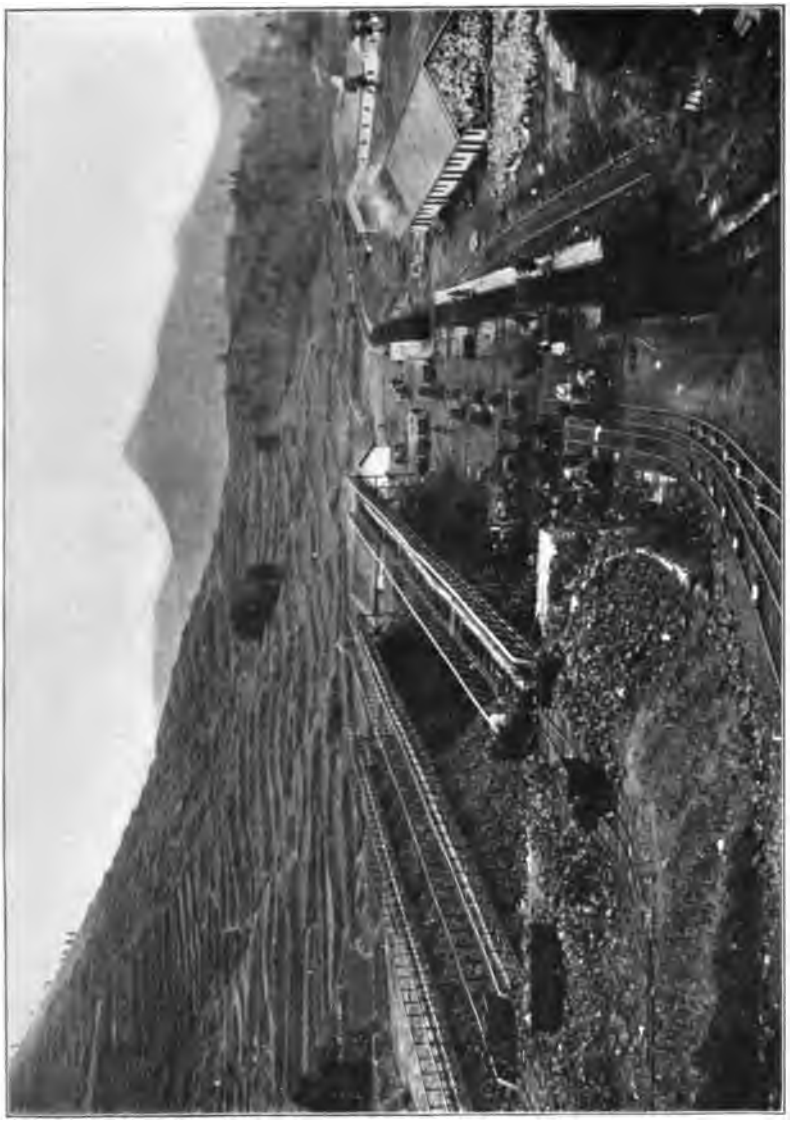

FIG. 15.—LOADING-PLANT AT TA-SHIH-MEN, TA-YEH IRON-MINES.

FIG. 16.—STRIPPING THE OVER-BURDEN IN THE OPEN-CUTS AT TA-SHIH-MEN, TA-YEH IRON-MINES. THE IRON-ORE IS HANDLED IN THE SAME WAY.

FIG. 17.—LOADING ORE IN CARS, TA-YEH IRON-MINES.

efforts were made to work this deposit as a copper-mine, but could scarcely have been very successful. The iron-ore is opened up over a length of more than 2 miles, but apparently is not continuous over that extent. The contact runs nearly east and west, and for considerable distances to each side of the present workings the ores can be traced. Where it is worked the ore-body is about 200 ft. thick and is nearly vertical, dipping slightly to the north. Fig. 15 is a view of the loading-plant at the Ta-yeh mines, and Figs. 16 and 17 illus-

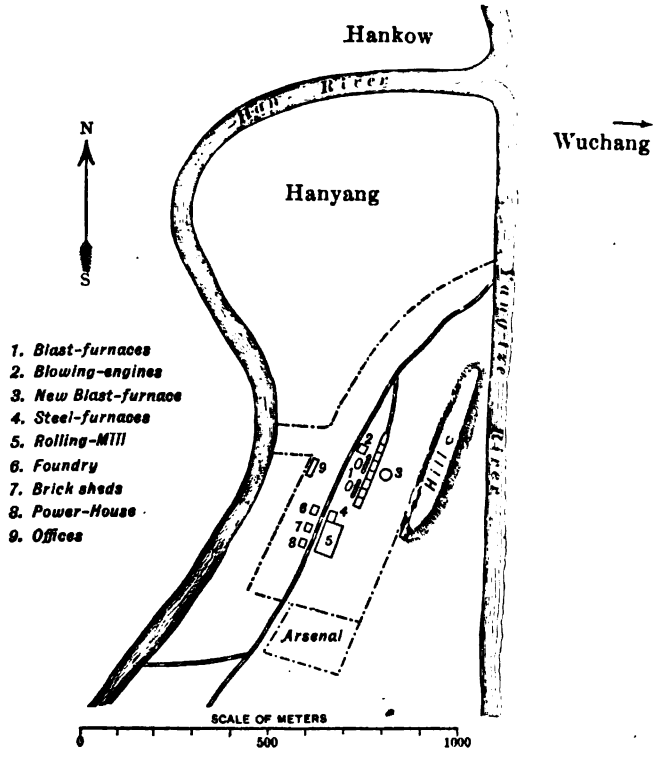

FIG. 18.—PLAN OF STEEL-WORKS, HANYANG.

trate the method of stripping the over-burden and loading the ore into cars. The syenite lies to the north of the marble, and a few miles farther north the marble again appears, iron-ore again being present along the contact. It would be entirely impossible to estimate the amount of ore available in this district, as in the workings only open-cuts are made, and the ore is nowhere blocked out in such a way that it can be accurately

measured; however, the officers of the mining company believe that they have many millions of tons of good ore on their property, and there seems no reason to doubt this.

At the metallurgical works at Hanyang, a plan of which is given in Fig. 18, two blast-furnaces, each of 100 tons daily capacity, are in operation, and a third unit of 250 tons capacity has just been completed. Three Siemens-Martin open-hearth furnaces are in operation, two are under construction, and five more are projected. The rolling-mill has a capacity of 800 tons per day; the larger part of it is devoted to rolling railway-material, as the greatest demand in China is for material of this nature. During 1909 there was produced 307,500 tons of iron-ore, of which a large part was exported to Japan and America, and the remainder smelted at Hanyang. Of the resulting pig-iron, 44,300 tons was sold as such, and the remainder converted into 28,500 tons of steel, chiefly steel rails; 3,600 tons of manganese-ore of good quality was produced in 1908 at P'ing-hsiang, in the province of Kiangsi. In several other places the company owns deposits of manganiferous limonite which contains about 20 per cent. of manganese, which can also be utilized if necessary. The company has a contract with the Japanese government iron-works at Wakamatsu to supply them with ore of Bessemer grade, and has recently made a contract with the Western Steel Corporation to furnish it yearly with 36,000 tons of ore and 36,000 tons of basic pig-iron.

The occurrence of iron-ore throughout the Empire is given in the following list of the places where deposits of iron-ore are known to exist, beginning at the north.

In Manchuria quite an amount of iron is produced [19] by native methods at T'ieh-ling, 44 miles north of Mukden, the ore coming from an adjacent range of hills. Recent Japanese reports are to the effect that iron-ores containing about 50 per cent. of iron exist along the line of the Mukden-Antung railroad, and also at Sai-ma-chi, Tung-hua, and 'Puaijin.' The NE. part of the province is only sparsely settled, and no other deposits have yet been opened.

Practically nothing is known of the iron-resources of the vast extent of Mongolia. At present the lack of transportation removes them from consideration, but it is not unlikely that

the Peking-Kalgan railroad will be extended to meet the Trans-Siberian road, and the coal- and iron-resources of this great area may have to be considered in the future.

The iron-ores of Shansi have already been described. Richthofen states that in Shensi the conditions as regards iron-ores are probably similar to those in Shansi, and assigns their lack of development to the character of the coal, which is not so suitable for the native methods of smelting. The fact that Shansi lies much nearer the markets for iron is also of importance, for where transportation is so expensive it would be impossible to ship across a producing district and compete with it, unless conditions were immensely more favorable in the more remote district. At any rate, Shensi must also be included as an area of which but little information is available at present, but which may become of importance in the future.

In the province of Chili iron-ore occurs at several places, notably in the NE. part; but I do not regard any of these as likely to become the basis of a permanent industry.

Magnetite occurs in the province of Shantung, at P'ao-shan, about 50 *li* south of T'ung-chou. Not far from this locality, at 'King-kwo-shan,' a different ore occurs, according to Williamson. [52] Near Chefoo specular ore occurs. It is not probable that the ores at these and numerous other places in the province offer much promise of success by development on a large scale, otherwise the German interests which have been so active in stimulating the growth of industry in the province would have undertaken their exploitation.

In Kiang-su province iron-ores are widely distributed, occurring chiefly in the region about Nanking in association with the limestone previously mentioned. Richthofen [45] seems to regard these of little promise because of the absence of coal of suitable quality in the neighborhood. But when the distance to which iron-ores are transported in the United States is considered, and the fact that the ores in question are conveniently adjacent to the Yangtze river, it seems much more probable that these ores will be the next to be developed on a modern scale. The ore is probably of similar character to that at Ta-yeh, previously described.

In the province of An-hwei the geological conditions are similar to those in Kiang-su and Hu-pei. A concession was

granted to the London & China Syndicate to exploit copper-mines in this province. A few years since, when the Chinese government was about to cancel the concession, the company claimed to have developed ore of a value of more than $4,000,000. But I have been unable to secure any accurate information regarding this concession.

The iron-ores of Hu-pei province have already been described. A recent report made by some students sent out by the provincial government is to the effect that workable deposits exist at six localities.

The province of Honan is largely alluvial plain. In the NW. part the conditions are similar to those existing in Shansi, and similar ores occur. These ores have not been the basis of a flourishing native industry, but some large deposits are known to occur, and it has recently been proposed to develop them by modern methods.

In Ssu-chuan iron-ore is widely distributed, and both Baber [1] and Hosie [18] mention numerous places in which it is the basis of a native industry; but neither of these authors gives enough details upon which to base a judgment as to the future of the industry.

In regard to Yunnan and Kweichou information is even more meager. Numerous travelers and explorers agree that iron-ores are widely distributed throughout Yun-nan, but there is an entire absence of definite information. A similar statement may be made in regard to Kweichou. The native industry in these provinces must be in a flourishing state, judging by the exports of native material.

On the maps prepared by La Mission Lyonnaise the following places are marked as productive of iron: Ssu-chuan (105° 50′ E., 32° 10′ N.), (105° 40′ E., 29° 45′ N.), (104° 40′ E., 29° 15′ N.), (104° 50′ E., 29° 05′ N.), (104° 30′ E., 28° 48′ N.), (102° 05′ E., 29° 40′ N.). Kweichou (103° 0′ E., 26° 10′ N.), (102° 50′ E., 26° 50′ N.), (103° 30′ E., 27° 15′ N.), (104° 0′ E., 25° 20′ N.), (104° 50′ E., 29° 0′ N.), (104° 50′ E., 29° 20′ N.), (106° 05′ E., 27° 0′ N.). Yunnan (99° 45′ E., 24° 20′ N.). Kwangtung (109° 20′ E., 24° 40′ N.), (111° 05′ E., 24° 25′ N.), (111° 05′ E., 23° 50′ N.), (112° 0′ E., 23° 45′ N.), (113° 30′ E., 23° 0′ N.). LeClere [25] says that iron-deposits are common in Yunnan, and are worked in places where the

supply of charcoal is abundant. He also refers to the iron industry in the northeastern part of Kweichou. Duclos [18] reports what is almost incredible, that the Chinese smelt iron-ores in blast-furnaces in Ssu-chuan.

Iron-ore of inferior quality occurs in Hunan, but is only worked by the natives at Chin-chou, near the southern border of the province, where the quality is much better. Very probably exploratory surveys would disclose valuable deposits at points not convenient for the application of native methods.

The manganese-ores at P'ing-hsiang, in the province of Kiangsi, have already been mentioned. Manganiferous limonites also occur along the Yangtze, in the NW. part of the province, and it is not improbable that ores occur in other parts of the province, but are not worked by the natives.

In Che-kiang province there is a native industry in an area extending from near Ning-po down into the province of Fu-kien. But as the ores consist of grains of hematite which are washed by hand from the sands of the streams, this district may be dismissed as of no future importance.

At 'An-Khoe,' in Fu-kien province, about 60 or 70 miles from Amoy, there is a large deposit of magnetite, estimated to contain 10,000,000 tons of ore, according to a report given to me by consular officials. It is favorably situated for working, but unfortunately the report says nothing as to the amount of titanium present in the ore.

In Kuang-tung and Kuang-si provinces, iron-ore is produced in Hsin-hui-hsien, and is both produced and smelted at Hsin-hui-hsien and Yang-an-hsien; the annual production at the latter place amounting in value to $250,000 gold.

The cost per ton of production of iron-ore in the open-cut workings at Ta-yeh is approximately as follows:

	Mexican Currency.
Stripping,	$0.08
Mining,	0.18
Tramming,	0.03
Powder, steel, etc.,	0.015
Superintendence,	0.06
Loading cars, freight to Yangtze, etc.,	0.30
Total,	$0.665

From these data the probable cost of the ore delivered at the blast-furnace would amount to a little more than $1, Mex., per

ton. The cost of limestone is about two-thirds of this; the cost of coke $15, Mex., and coal about $8, Mex. These low costs are largely due to the low cost of labor. Ordinary unskilled labor receives from 200 to 300 *cash* per day, or from $0.08 to $0.12 gold. Skilled labor receives from $5 to $25 gold per month, and the efficiency of this labor is remarkably high. As a result, pig-iron and steel can be produced at a very low cost; pig-iron from this plant is not only sold in Japan, but also in New York and San Francisco. But as there is a growing market for the present output in China, it is scarcely likely to become a serious competitor in other markets, at least for the present.

In conclusion, it may be said that, while our knowledge of the iron-resources of the Chinese Empire is still inadequate, yet the general features appear to be these: Iron-ores are widely distributed throughout the Empire, from north to south. In many places these have been the source of a more or less considerable native industry, which is steadily waning, because of the unsatisfactory quality of the product. Owing to the peculiar features of the native methods it is not to be supposed that modern industries will necessarily develop in the same localities; on the contrary, the one modern plant is utilizing ores not previously worked by the natives, and this will not improbably be the policy of its successors. In this development of the iron and steel industry it is most probable that the Yangtze valley will have a leading place, as there seems good reason to believe that iron-ores of satisfactory quality, and in sufficient quantity, occur along a considerable portion of its length, and it also possesses the great commercial advantage of easy and cheap transportation, and a situation in the commercial center of the Empire. The extremely low cost of labor permits a low cost of production, making tariff for the protection of the industry entirely unnecessary. The low price of silver in recent years has operated to benefit China in a twofold manner: both by stimulating export trade and by discouraging imports. The resultant increase of wealth will give increased ability to meet the initial large expenditures necessary to develop modern plants. A large and increasing market is assured by the Imperial regulation that all materials required in the construction of railroads and other public works shall

be purchased from the Chinese plants, so far as these are able to furnish them. As a result, it is to be expected that the iron and steel industry of China will have a large and healthy growth.

IV. Gold.

Gold in China has unusual interest at this time, since the proposal to place the coinage of the Empire upon a gold basis is now receiving more serious consideration. The fact that the gold-production is so small has always acted as a strong deterrent to the establishing of a gold coinage. Since China produces but little of either gold or silver, and must purchase them with exports of merchandise, the advantage of buying silver at its present low price is apparent. At present silver and copper are the media of exchange; the former passing at an approximation to its bullion value, while the copper *t'ung-tz'erh* and *cash* are subject to daily fluctuations of value. As each province now issues coins, which pass only at a discount, or perhaps not at all, in other provinces, the transaction of business is subjected to a wholly unnecessary burden. The central government, having acquired control of the railways, will doubtless next proceed to complete its present shadowy control of banking, and a gold standard may be introduced, as large amounts of foreign capital are now invested in China, and interest payments are at present subject to sudden and extensive changes in the rate of exchange.

The absence of records of mineral production renders it difficult to obtain any knowledge of the gold-production of the Empire. The best approximation can be made from the customs-records of gold exported, since the amount used annually in the arts is but small. The export for 1908 was approximately $6,500,000; that for 1907 was $3,200,000. The occurrence of gold in China has been noted by Baber, [1] Garnier, [15] Duclos, [13] LeClere, [25] Jack, [21] Verschoyle, [49] Hoover, [16] Hosie, [18], [19], and others, [9], [14], [20], [26], [27], [28], [29], [33], [34], [35], [36], [39], [43], [44], [50], [51], [52], [54], [57]; but the most notable descriptions are by Hoover, [16] Hosie, [19] and Purington. [36] The last has described at length the occurrences of alluvial gold in northern Manchuria, along the Amur, Sungari, Tumen, Urga, and Nonni rivers and their tributaries, and its recovery by primitive washing-methods.

Gold also occurs in southern Manchuria, both in alluvial deposits and in narrow veins, frequently of the gash type. The Bureau of Mines of Manchuria has compiled a list of 10 localities where gold is worked in Fengtien province, and 40 others where it is known to occur.

Bogdanovitch [2] has published an excellent study of the deposits of the Liao-tung peninsula, dividing them into four classes: (1) existing stream-beds; (2) Pleistocene high-level gravels; (3) ancient valley alluvials; (4) marine placers. The northern area is of much greater importance. It will perhaps be safe to estimate that Manchuria produces from 75 to 80 per cent. of an annual output of from $5,000,000 to $7,000,000 for the whole Empire.

The gold-deposits of Chili have been described by Hoover. [16] These are widely scattered, the most notable being veins at Chin-Chang-K'ou (119° 56′ E., 42° 20′ N.), and at Chuan-Shan-tze (119° 12′ E., 42° 26′ N.). In 1910 the former was producing at the rate of $150,000 per year, and the latter $15,000 annually. There are numerous placers throughout the province, and Hoover [16] estimated the total production for 1908 as $1,000,000. In Shantung similar conditions prevail, except that the deposits are not so abundant.

The best-known mine in China is that of Chou-Yuen, about 40 miles SW. of Chefoo, where a quartz vein from 40 to 90 ft. wide has been uncovered for more than a mile in length. Curle [9] reports that 200,000 tons of ore, worth $10 per ton, has been developed. The ore is about 40 per cent. free milling, and many years ago a chlorination-plant was in operation upon the tailings from the stamps. The mine has been closed for many years by government order; but there has been constant effort on the part of foreign companies to secure the concession, and recently the Governor of Shantung petitioned the Peking government to allow the Chinese owners to operate the mine, fearing lest it might otherwise be lost. The near-by mines at P'ing-tu are probably of considerable value. In spite of the government prohibition of mining a certain amount of work is carried on quietly, and Hoover [16] estimated the gold-production of Shantung at $12,000 annually. With the few exceptions noted there seems little probability that deposits of sufficient size can be developed to justify the construction of modern

milling-plants, the cost of labor being so low that primitive methods give cheaper working-costs. The tailings from the chlorination-plant, mentioned above, were bought by the neighboring farmers, who carried the material home, and, in the dull agricultural season, ground it in native mills and panned it. The deposits near Wei-hai-wei have been described by Verschoyle, but are of little importance. In northern Manchuria, which has not been adequately prospected, large-scale workings may perhaps be developed; but throughout the rest of the Empire progress is likely to take the form of increase and improvement of native mines, efficient and inexpensive pumps being the most-needed equipment of native mines. Hansen has recently reported that between Lanchow, on the western border of Kansu, and the border of Thibet, numerous placer-workings exist, carrying as much as 2 g. of gold per cubic yard. A modern gold-milling plant is in course of erection at this place. Quartz veins also occur, containing from 1.5 to 2 oz. of gold per ton; but as these statements are apparently based upon the reports of natives, they must be received with caution. A recent Russian report is to the effect that on the areas of the Tushetvohanovsky and Tzentzenhanovsky concerns, Mongolia, between January 14 and September 2 this year, the gold-production was 84.25 poods (1 pood of placer gold equals $9,000 approximately), as compared with 88 poods in the corresponding period of 1910.

Throughout the rest of the Empire quantities of gold are produced in many places, most of which are known to be of little importance. Ssu-chuan and Yunnan have been of much interest, since they are but little known. The upper waters of the Yangtze are known as the Chin-sha, or 'golden sand,' but this does not necessarily indicate, as has been assumed, that especially rich alluvials exist along its course. Many travelers have mentioned the known or rumored existence of gold-deposits in these two provinces, LeClere, Duclos, Baber, Hosie, Garnier, Jack, and Johnston, to name but a few, but definite information is only meager. At Mo-Lo, in Ssu-chuan (102° 05′ E., 28° 15′ N.), important quartz-deposits have been worked since 1880 by a combination of foreign and native methods. Small quantities of alluvial gold are reported from a great number of places, but in nearly every case the name of the

district is given in such a way that identification is difficult or impossible. In giving the native names of little-known places, either the Chinese characters should be employed, or else the latitude and longitude should be given. In Yunnan the most notable mines are at Ta-lan-tung (101° 45′ E., 23° 30′ N.) at an elevation of 7,300 ft. (LeClere[25]). These quartz veins in Palæozoic rocks are worked by native methods, yielding some $60,000 per year. At Kin-Kiang, 60 miles from Ta-li-fu, rich conglomerate beds are worked by the natives. On the maps prepared by La Mission Lyonnaise the following places are marked as productive of gold: Ssu-chuan (99° 36′ E., 30° 55′ N.), (99° 45′ E., 30° 40′ N.), (99° 55′ E., 30° 20′ N.), (101° 50′ E., 32° 40′ N.), (101° 40′ E., 32° 0′ N.), (102° 15′ E., 31° 40′ N.), (102° 0′ E., 29° 0′ N.). Yunnan (98° 45′ E., 23° 45′ N.), (99° 50′ E., 23° 20′ N.). Numerous other localities are mentioned in the references given, and with the great recent increase of transportation-facilities in Yunnan it is not at all impossible that the gold-mining industry will greatly develop.

Gold is known to occur in Kansu, and there are quartz veins in the upper valley of the Han-ho, in Shensi. Along the lower reaches of the Han-ho, in Hu-pei, there is a small amount of gold-washing constantly carried on. The same is true of Fukien, where, in the Shao-wu district, 150 miles NW. of Foochow, valuable deposits are said to exist. In Anhwei mines were formerly worked, and deposits are known to exist in Hunan and Kwangsi, but they are not much worked. The mines in Kwangtung are said to be valuable; but in all these cases, it is probable that the deposits are limited in extent and of no great richness. It may be said, in conclusion, that with the exception of Manchuria and the SW. provinces of China (Yunnan, and Ssu-chuan), the gold-mining industry gives but little promise of growth. In the districts mentioned the introduction of modern pumping-machinery and the removal of the restrictions which the authorities and superstition have placed upon the industry are likely to lead to a considerable development.

V. Silver.

Silver is of great interest in China, since it is the chief medium of exchange. But the domestic production of silver

is small, and China is, accordingly, a heavy buyer of the metal. An interesting subject for speculation is found in the source of supply of this metal during the period when China maintained but little communication with the outside world. About the time of the beginning of the Christian era there was a great deal of traffic with the countries on the SW. border, where the mining and smelting of argentiferous galena seems to have been a considerable industry. The best-known silver-mines are those in the northern part of Chili province, near Jehol. At the time when Li Hung-Chang was viceroy, foreign engineers were employed in their exploitation. Woo, [55] who has described the native methods of mining and smelting these silver-lead ores, which occur as veins in quartz-porphyry associated with Palæozoic sediments, says that the lack of success of the foreign enterprise was due to the high cost of the coke used and to the unsuitable character of the milling-machinery employed. The general and probably correct impression is that these veins are too limited in extent and too irregular in character to allow of their being worked with a profit upon a larger scale than that employed by the natives. The production in 1903 was estimated, [55], [16], at from 80,000 to 100,000 oz. per year. In the mountains to the west of the line of the Peking-Hankow railroad numerous small occurrences are known, but the production is probably insignificant. References to the occurrence of silver-ores in nearly every province may be found, but do not seem to warrant much attention. Duclos [13] mentions numerous places in Yunnan where silver is produced, being usually associated with lead and zinc. The production at Pei-cha-po he estimated as 10,000 oz., and says that small amounts are produced at 'Koung-chan,' 'Fou-lay-tchang,' 'Siao-in-tchang,' 'Tchou-tsin-tchang,' 'Sin-pao-tong,' 'Ta-lang-tchang,' 'Tchen-pien-tchang,' 'Mo-lay-tchang,' and 'Sin-tchang,' in Yunnan. At 'Tsai-tse-chang,' in Kweichou, silver is obtained from an argentiferous galena. At 'Mou-pin,' in Ssu-chuan, an argentiferous galena containing a little gold occurs; silver is also found towards the north of the province, at 'Pe-tsoa-pa' and 'Hou-koua-tou.'

On the maps prepared by La Mission Lyonnaise the following places are marked as productive of silver: Ssu-chuan (101° 45′ E., 32° 10′ N.). Kweichou (104° 20′ E., 25° 12′ N.),

(104° 35′ E., 25° 05′ N.), (104° 15′ E., 28° 10′ N.). Yunnan (98° 45′ E., 23° 45′ N.), (99° 50′ E., 23° 20′ N.), (101° 0′ E., 26° 30′ N.). Kwangsi (106° 55′ E., 23° 0′ N.), (107° 50′ E., 21° 30′ N.). Kwangtung (108° 20′ E., 21° 40′ N.), (109° 45′ E., 22° 20′ N.), (110° 50′ E., 25° 10′ N.), (111° 0′ E., 24° 50′ N.), (111° 05′ E., 22° 20′ N.), (112 15′ E., 22 50′ N.), (114° 05′ E., 23° 25′ N.), (114° 05′ E., 24° 15′ N.). A Chinese company is now mining silver at Kwei-Hsien, in Kwangsi. The Jun Wah Sut Yip Co. was organized a few years ago with native capital to develop silver- and other mines in Kwangsi, and is carrying on extensive exploration- and development-work. Several foreign companies have attempted the development of silver-mines in southern China, but without much success. It must be admitted that, on the whole, silver-mining in China is of no great importance.

VI. Copper.

Copper has always been an important metal in China, brass and bronze, used for objects of use and adornment and as media of exchange, having played a prominent part in the national life from the earliest times. But in the case of copper, tin, nickel, zinc, and lead, the discussion of the metals separately offers difficulties, since the ores are associated and are often smelted together, giving rise to natural alloys. By far the most important supply of these metals is derived from the southwest. Yunnan is the most important, but copper also occurs in the neighboring regions. Duclos [18] gives a long list of places where copper is produced in Yunnan, and describes in some detail the work at San-Kia-Chang (99° 35′ E., 24° 40′ N.), where he estimated a yearly production of 42 tons. LeClere [25] has summarized as follows:

"The total production obtained by reduction with charcoal was, in the 17th century, at least 6,000 tons, but has decreased to from 1,000 to 1,500 tons. The decrease is not due to the impoverishment of the deposits, but to the disappearance of the forests, the scarcity of fuel permitting only the exploitation of deposits of unusual richness at the places most favorably situated in respect to transportation. The copper minerals are various. Phillipsite, cantonite, and more rarely cupriferous pyrite, are found in veins in the Carboniferous schists, but are almost abandoned. Sheets of cuprite with barite, and of native copper, intercalated in porphyrite, are highly esteemed, but the lack of explosives often makes it impossible to work them. Sandstones impregnated with copper carbonate are often found in the Trias. The principal deposits are in Triassic limestones, the net-work of veins

have been converted into carbonates, with only a trace of pyrite. The principal mining centers are: (1) 'Toung-tchouan' (101° 0′ E., 26° 30′ N.), the mines in Ssu-chuan, near the Blue river are connected with these; (2) 'Oui-si,' near Li-Kiang (97° 15′ E., 27° 15′ N.); (3) the neighborhood of 'Ouei-ning' (102° 15′ E., 26° 45′ N.) in Kweichou produces zinc and lead principally, and is directly governed by the mining official of Yunnan-hsien. The works will not take ores of lower grade than will give a smelting mixture of from 20 to 30 per cent. of copper; the latter is more common. This richness is obtained by careful hand-sorting at the mines. The ores lower in grade than 15 per cent. are piled up and form large dumps. The old slag-dumps are common near the old smelting centers and contain about 3 per cent. of copper. These facts demonstrate that Yunnan has extremely large resources of copper minerals, especially in the parts not workable by native methods."

The present chief source of supply from Ssu-chuan, according to Fox,[58] is at Hwei-li-chon, though an important mine exists at Pai-shui-ho.

Copper occurs in many other parts of China. While in Kiangsi in 1908, the local officials told me of a large copper-mine at Chang-pai-ling in the west-central part, which had recently been extensively developed, but was not successful. Clark[4] has described the copper-ores at Lao-Pao-chi in Anhwei; this is probably the area which was subsequently granted as a concession to Lord Lister Kaye. Considerable work was done upon this concession by the English company, but without conspicuous success, and in 1910 the concession was surrendered. Clark[4] also describes the production of copper by native methods near Jehol (118° E., 40° N.), and I have seen unimportant native workings in many other places. C. H. Hansen, who has recently been engaged in the work of constructing a 50-ton copper-smelting plant under government auspices at Yaokai, 70 miles west of Lan-chow, on the western border of Kansu, says that there are numerous mines within a radius of 80 miles from that point.

The mining of copper has always been rigorously under the control of the government, probably because of the importance of the metal as a medium of exchange. The native supply has not been equal to the demand recently, and considerable amounts are imported, chiefly from Japan.

VII. Nickel.

Nickel is of much technical interest because of the ingenious way the Chinese have of smelting mixtures of nickeliferous

copper-ores with tin, lead, and zinc-ores, forming the alloy "pai-t'ung," or "pakfong," as it is called in southern China. This is a kind of German silver, which is extensively used in the manufacture of candlesticks and other household objects. Nickel is never produced separately, and the entire supply is apparently drawn from southwestern China, where in Yunnan (at 100° 20′ E., 26° 50′ N.), and in Ssu-chuan (100° 20′ E., 26° 45′ N.), Duclos [13] has noted the occurrence of nickeliferous copper-ores.

VIII. Tin.

China is an important producer of tin, furnishing at present about 5 per cent. of the total production of the world. A certain amount of the product comes in native boats down the Yuen and Hsiang rivers, probably originating at some point in Kweichou. Kwangtung also produces a notable amount, which is exported through Wuchow. On the maps prepared by La Mission Lyonnaise the following places in Kwangsi and Kwangtung are marked as productive of tin: (105° 10′ E., 24° 12′ N.), (110° 15′ E., 24° 45′ N.), (110° 45′ E., 32° 15′ N.), (112° 0′ E., 23° 30′ N.), (112° 45′ E., 22° 50′ N.), (114° 10′ E., 23° 40′ N.). There are also deposits in Fukien, and a small amount is shipped from Foochow. But by far the larger part of the production comes from the well-known mines at Ko-ch'iu-ch'iang (100° 50′ E., 23° 20′ N.), in Meng-tze-hsien, Yunnan. Here the deposits, which LeClere [25] says "ne sont nullement alluvionnels: leur origéne filonienne est des plus evidentes," are scattered over an area 25 miles long and 20 miles broad, and 30,000 workers are engaged in mining and smelting the product of 150 mines. The workings are both surface and underground and only native methods are employed. Collins, [6] Duclos, [13] and LeClere, [25] have described these deposits in detail. Theoretically the deposits belong to the central government, which, according to the mining-regulations, exacts a royalty of 25 per cent., but the amount actually paid is somewhere between 12.5 and 18 per cent. The reports as to the production of these mines do not agree, but the customs return for 1909 was 4,700 tons, corresponding to about 90 per cent. of the total production for the Empire. The crude tin, in slabs, each weighing 74 lb., is exported to Hongkong, where it is refined. A large part of the tin was formerly

shipped down the Yangtze and West rivers, but now it is shipped over the French railroad. Recent newspaper reports are to the effect that efficient smelting- and mining-machinery is to be installed, about $1,200,000 having been subscribed for the purpose, and the erection of a slag-cleaning plant has been proposed. It is not unlikely, therefore, that the production of tin in China will considerably increase in the near future.

IX. Lead.

There is a considerable demand for lead in China, which is largely met by imports, 10,707 metric tons of pig-lead having been imported in 1908. The domestic production is not inconsiderable, lead-ores occurring in 8 of the 18 provinces; Kweichou, Ssu-chuan, and Yunnan are the most important; but about 1,500 tons of ore were derived from Hengchou and Yangchou prefectures, in Hunan province, in 1908. The larger part of this is exported to Great Britain and Belgium, but considerable quantities are smelted at the works of Carlowitz & Co. at Wu-chang. About 350 tons of pig-lead was brought down the Yangtze through I-chang in 1907, and 300 tons in 1909. A good deal of lead-ore comes down the Yuen and Hsiang rivers, both from Hunan and Kweichou. Duclos [13] has described the production of lead at 'Tcha-tze-tchang' (102° 50′ E., 26° 45′ N.), in Kweichou, and notes its occurrence at (102° 40′ E., 26° 30′ N.). In Yunnan the mines contain mixed ores of lead, zinc, copper, and silver. In the neighborhood of 101° 0′ E., 26° 30′ N., at 'Kong-Chan-tchang' 1,450,000 lb. of lead is produced, at 'Pe-cha-po', 700,000 lb. of lead, at 'Koung-chau' the lead produced is cupelled for silver. Near 102° 20′ E., 26° 45′ N. there is a considerable production, both of lead and zinc. In many places argentiferous lead-ores are worked (see "Silver"), the bullion being cupelled; the resulting litharge often being thrown away. LeClere [25] estimated the annual production in southwestern China at around 3,000 tons. The lead-ores of Kwantung and Kwangsi are mentioned under "Silver." On the maps prepared by La Mission Lyonnaise the following points in Kwangtung are marked as productive of lead: (109° 50′ E., 21° 58′ N.), (110° 15′ E., 23° 57′ N.), (114° 05′ E., 23° 58′ N.). The native methods of metallurgy are so imperfect that it is not improbable that in many cases

deposits exist which could be worked by modern methods. But, as mentioned under "Silver," it has not infrequently been found that the veins are too narrow and irregular to yield a profit when exploited on a large scale.

X. Zinc.

The whole domestic supply of zinc in China is derived from the SW. provinces, chiefly Kweichou. Duclos [13] has described the native metallurgical methods followed at 'Ma-lou-kio' (102° 50′ E., 26° 45′ N.), and mentions the occurrence of zinc in Yunnan at (102° 20′ E., 26° 45′ N.) and (101° 40′ E., 25° 30′ N.). LeClere [25] says that the workings are only superficial, and that the distillation-methods are so imperfect that much ore which could be utilized by modern methods has been discarded. He estimated the annual production at about 2,500 tons. The spelter from Kweichou, which I have seen, was obviously impure, but it is used in the provincial mints in making brass coins, without any attempt at refining. Large quantities of zinc-ore are produced in Hêng-chou and Yang-chou prefectures, Hunan, and exported to Belgium and Holland. The export for 1908 was more than 15,000 tons, but for 1909 was very small.

XI. Antimony.

China possesses the distinction of being first in the production of this metal. The condition of the industry has been changing rapidly of recent years. My notes were made on a visit to the chief center of production in 1908. The ore comes from a number of places to NW of T'ung-t'ing lake in Hunan, I-yang being the chief center (110° E., 29° N., approximately). It is carefully hand-sorted at the mines, and is brought in native boats to Changsha (112° '45′ E., 28° 15′ N.), where it is liquated in pots about 15 in. in diameter. The methods were entirely Chinese at that time, but I understand that furnaces of modern construction have since been installed. The regulus is sent to Hankow, and so much as desired is converted into metal. The residues have been exported to France, Germany, and the Netherlands for further treatment; more recently large quantities have been sent to the United States. Carlowitz & Co., who operate a smelter at Hankow, are also engaged in the work of smelting and refining both antimony

and lead, but as strict secrecy is observed, details are not obtainable. The production for 1907 from this district in Hunan, according to customs returns, was 3,957 tons of regulus and 14,810 tons of ore. In 1908 the production of regulus increased to more than 8,000 tons, and the ore-shipments decreased to less than 2,000 tons. The native company which operates the smelter has trained metallurgists in its employ, and the industry is likely to further increase in importance.

Antimony also occurs in Kiangsi, Kweichou, and Ssu-chuan, but Kwangtung is the next most important province after Hunan. Recently a smelter under government auspices was established at Wuchou (111° 30′ E., 23° 30′ N.), but was not a success, and has since been closed. A more extended discussion can be found in the *Mineral Industry* for 1908 and 1909, and in the volume *Antimony*, by C. Y. Wang (London, 1909).

XII. Quicksilver.

Quicksilver has a steady demand in China, most of the metal being used in the production of vermilion, which enjoys high favor as a pigment, a less important use being in gilding by the Chinese method. The native production fluctuates greatly, the amount reported by the customs authorities being 30 tons in 1907, and 65 tons in 1908. It is impossible to determine what relation exists between the total amount produced and that reported by the customs. Most of the demand is supplied by imports, largely from California. The occurrence of quicksilver is reported from many parts of the Empire, but the deposits in the province of Kweichou are, apparently, the only ones of commercial importance. As I have not visited these districts, my information is entirely derived from published statements. Duclos[13] has noted several places in Kweichou where quicksilver is produced; 'Oa-tchouan' (105° 50′ E., 28° 30′ N.), Pei-ma-t'ong (104° 15′ E., 26° 48′ N.), Lan-mou-chang (103° 15′ E., 25° 28′ N.), and Yang-li (104° 20′ E., 15° 0′ N.) are the most important. He estimated the production at Pei-ma-t'ong as 6,500 lb. per year, and says that at Yang-li there is "*une enorme exploitation.*" He also mentions the occurrence of quicksilver in Yunnan (101° 10′ E., 23° 30′ N.). LeClere[25] was not favorably impressed by the mercury-deposits, and remarks, accurately enough, that the native methods of work can

treat ore of quite as low grade as would be profitable by foreign methods. Brelich[3] says that the deposits at Wan Shan Chang, in Toon Yen prefecture, are the largest and most extensively worked in Kweichou. They occur in nearly horizontal beds of magnesian limestone: (1) impregnating well-defined beds; (2) along joints and planes of stratification; (3) in isolated bunches and vugs; (4) irregularly disseminated. The ore is hand-sorted and retorted in native furnaces. Brelich says the miners work for daily wage, which is unusual. The Anglo-French Quicksilver & Mining Co. began work at Kwei-yang in 1899, with a capital of £310,000, operating the Wen Shan Chiang mines, which are 12 miles north of the Yuen river. These are apparently the mines which Brelich describes. The area of the concession was 4 sq. miles. Two 12-ton Granitza furnaces were constructed, and the production from 1899 to the end of 1902 was 32,500 lb. of quicksilver and 500 lb. of cinnabar. The company has had a somewhat checkered career, and little progress is now being made.

XIII. Arsenic.

Arsenic is of no little importance in China, but detailed and accurate information regarding its occurrence and production is not obtainable. According to the customs returns, 5,000 tons was produced in 1908, but only 400 tons in 1909. Apparently the ore occurs in or near the principal antimony district (111° E., 29° N.), since the product passes through the same custom-house. About 700 tons of arsenic-ore from China was imported into Germany in 1908. A recent French report estimates the annual production of orpiment and realgar in Yunnan at 600 tons. This comes from near Chao-chou and Meng-hua, in Tali prefecture.

XIV. Petroleum.

The most notable petroleum-producing district is in Ssu-chuan, where, according to Coldre,[5] there are in the neighborhood of Yun-hsien (102° 10′ E., 29° 38′ N.) and Fu-chuan (102° 43′ E., 29° 27′ N.) from 30 to 40 wells, from 1,000 to 3,500 ft. deep, in which petroleum occurs associated with gas and brine. The gas is employed in evaporating the brine, and the petroleum, which varies from '*blanc de petit lait*' to '*noir*,' is burned in crude lamps without any attempt at refining.

Duclos[42] describes the wells at 'Tse-liou-tsin,' which is between Yun-hsien and Fu-chuan, nearer the latter, giving the number as about 150; but probably this refers to the total number rather than those which actually produce petroleum. But neither gives any estimate of the amount of actual production. Richard[43] refers to the occurrence of petroleum in Kansu, but does not give the locality. Recent press reports are to the effect that petroleum has been found near the Kanchou coal-mine, in Kansu, but no definite information is available.

A field that is of present importance occurs near Yen-chang (110° 0′ E., 36° 30′ N.), in Shensi. A native company has been at work there for some years, has recently constructed a refinery, and is now selling oil throughout Shensi in active competition with the foreign product. I have been unable to obtain reliable detailed information regarding this field, but it is probably of a great deal of importance, since it is rumored that the company is considering the construction of a light railway from the wells to Singan-fu, 200 miles distant. Apparently petroleum occurs only in the western and SW. parts of the Empire, but in these places a considerable native industry is likely to develop when the areas are provided with adequate transportation-facilities.

In concluding this paper, it should be added that the development of the mineral resources of the Chinese Empire would be greatly aided by the creation of an Imperial Geological Survey. The nucleus of such an organization already exists in the many Chinese geologists and engineers who have been trained abroad, and with proper organization and sufficient funds, this important and necessary work would redound to the immense benefit of the mineral industry.

XV. Bibliography.

[1] Baber, E. Colborne: Travels and Researches in Western China, *Royal Geographical Society Supplementary Papers*, Vol. I, Pt. I.

[2] Bogdanovitch: Materialen zur Geologie Russlands, *Zeitschrift für praktische Geologie*, Vol. VII, p. 240 (1899).

[3] Brelich: Chinese Methods of Mining Quicksilver, *Transactions of the Institution of Mining and Metallurgy*, Vol. XIV, p. 483 (1904–05).

[4] Clark, Ellis: Notes on the Progress of Mining in China, *Trans.*, XIX, 571 (1890–91).

[5] Coldre, Louis: Les Salines et les Puits de Feu de la province du Se-Tchoan, *Annales des Mines*, Ser. 8, Vol. XIX, p. 441 (1891).

[6] COLLINS, W. F.: Tin Production in the Province of Yunnan, China, *Transactions of the Institution of Mining and Metallurgy*, Vol. XIX, p. 187 (1909–10).
[7] COLLINS, W. F.: *Mineral Industry*, Vol. XVIII, p. 672 (1909).
[8] CORDIER, HENRI: *Bibliotheca Sinica* (1904).
[9] CURLE, J. H.: *Gold-Mines of the World* (1899).
[10] DRAKE, N. F.: The Coal-Fields around Tsê-Chou, Shansi, China, *Trans.*, XXX, 261 (1900).
[11] DRAKE, N. F.: The Coal-Fields of Northeastern China, *Trans.*, XXXI, 492 (1901).
[12] DUCLOS: *La Mission Lyonnaise d'exploration commerciale en Chine*, 1895–7. Pp. 283 to 314 (Lyons, 1898).
[14] GARRISON, F. LYNWOOD: *Mining and Metallurgy*, Feb. 15, 1901, p. 106.
[15] GARNIER: *Voyage d'exploration en Indo-Chine*, Vol. I, p. 230 (1867).
[16] HOOVER, H. C.: Metal Mining in the Provinces of Chi-li and Shantung, China, *Transactions of the Institution of Mining and Metallurgy*, Vol. VIII, p. 324 (1899–1900).
[17] HOOVER, H. C.: Kaiping Coal Mines and Coal Field, Chihle Province, North China, *Transactions of the Institution of Mining and Metallurgy*, Vol. X, p. 419 (1901–02).
[18] HOSIE, ALEXANDER: *Three Years in Western China.*
[19] HOSIE, ALEXANDER: *Manchuria, Its People, Resources and Recent History.*
[20] HUC, E. R.: *Travels in Tartary, Tibet, and China.*
[20a] INOUYE, K.: *Geology of the Southern Part of Shêng-ching Province* (Tokyo, 1905).
[21] JACK: *Back Blocks of China.*
[22] JOHNSTON: *From Peking to Mandalay.*
[23] JAMESON, C. D.: Coal and Iron in Eastern China, *Engineering and Mining Journal*, Vol. LXVI, p. 365 (1898).
[24] KWANG, K. Y.: The Kaiping Coal-Mine, North China, *Trans.*, XVI, 95 (1887–88).
[25] LECLERE, M. A.: Étude Géologique et Minière des Provinces Chinoises voisines du Tonkin. Paris, 1902. *Annales des Mines*, Ser. 9, Vol. XX, p. 287 (1901).
[26] LE PRINCE-RINGUET: Étude Géologique sur le nord de la Chine, *Annales des Mines*, Ser. 9, Vol. XIX, p. 393 (1901).
[27] LORENZ, TH.: Beitrage zur Geologie von Ostasien, *Deutsche Geologische Gesellschaft*, Vol. LVII, p. 475 (1905).
[28] MACLAREN, MALCOLM: *Gold*, p. 270 *et seq.* (1908).
[29] MARGARY, A. R.: *Journey from Shanghai to Bhamo* (1876).
[30] MOLLER, W. A.: The Fushun Colliery, South Manchuria, *Trans.*, XLI, 241 (1910).
[31] MONOD, G. H.: Coal Fields of Indo-China, *Revue Métallurgique* (Sept. 19, 1904).
[32] MURDOCH, J. V. B.: Brine and Oil Wells in Western China, *Transactions of the Institution of Mining and Metallurgy*, Vol. IX, p. 362 (1900–01).
[33] PERVINQUIÈRE: *Revue Scientifique*, Ser. 5, Vol. I, p. 547 (1904).
[34] PREJEVALSKI: *Travels in Mongolia.*
[35] PUMPELLY, RAPHAEL: Geological Researches in China, *Smithsonian Contributions*, Vol. XV, p. 118.
[36] PURINGTON, C. W.: Northern Manchuria, *Mining Magazine*, Vol. IV, p. 53 (1911).
[37] READ, T. T.: Coal Mining in China, *Mining and Scientific Press*, Vol. XCVIII, p. 44 (1909).

[38] READ, T. T.: Coal Mining in Manchuria, *Mining Magazine*, Vol. I, p. 215 (1909).

[39] READ, T. T.: Mineral Resources of Manchuria, *Mining Magazine*, Vol. II, p. 121 (1910).

[40] READ, T. T.: Steel Making in China, *Mining Magazine*, Vol. II, p. 199 (1910).

[41] READ, T. T.: Iron Ore Resources of the Chinese Empire, in *The Iron Ore Resources of the World*, pp. 915 to 924 (Stockholm, 1910).

[42] The Lanchow Mining Co., Ltd. *Far Eastern Review*, Vol. VIII, p. 41 (July, 1911).

[43] RICHARD, L.: *Comprehensive Geography of the Chinese Empire* (Shanghai, 1908).

[44] RICHTHOFEN, F. F. VON: *China, Ergebnisse eigener Reisen und darauf gegründeter Studien* (1877).

[45] RICHTHOFEN, F. F. VON: Letters to the Shanghai Chamber of Commerce, 1872 (Shanghai, 1903).

[46] ROCHER: *La Province Chinoise du Yunnan* (1879).

[47] SHOCKLEY, W. H.: Notes on the Coal- and Iron-Fields of Southeastern Shansi, *Trans.*, XXXIV., 841 (1903).

[48] SWENSEN, K. P.: The Pinghsiang Colliery, *Mining and Scientific Press*, Vol. CI, p. 564 (1910).

[49] VERSCHOYLE, W. D.: Gold Mining at Wei-hai-Wei, China, *Engineering and Mining Journal*, Vol. LXXXII, p. 919 (Nov. 17, 1906).

[50] VOGELSANG: *Petermanns Mittheilungen*, Vol. XLVII, p. 245 (1901).

[51] VON CHOLNOKI: *Földtani Közloni*, Vol. XXIX, p. 289 (1900).

[52] WILLIAMSON, ALEXANDER: *Journeys in North China, Manchuria, and Eastern Mongolia*, 1870.

[53] WILLIS, BLACKWELDER, and SARGENT: *Research in China*, 1907. (Published by the Carnegie Institution.)

[54] WILLIS, BAILEY: The Mineral Resources of China, *Economic Geology*, Vol. III, pp. 1 and 118 (1908).

[55] WOO, Y. T.: Silver-Mining and Smelting in Mongolia, *Trans.*, XXXIII, 755 (1902).

[56] WOO, Y. T.: The Manufacture of Coke in Northern China, *Trans.*, XXXVI, 661 (1905).

[57] YULE, SIR HENRY: *The Book of Ser Marco Polo, the Venetian* (1875).

[58] FOX, H. H.: Memorandum on Chinese Mines, *Diplomatic and Consular Reports*, No. 680 (London, 1911).

The Mining Industry of Japan.

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(San Francisco Meeting, October, 1911.)

I. Before the Restoration.

1. *First Period.—From Remote Antiquity to 1000 A.D.*

At a time of great antiquity when our Yomato tribe had not yet found its way throughout the country, there lived in Japan barbarous tribes of the stone age, whose dwellings were vertical caves covered with roofs of twigs and weeds. In a cave lately discovered at the village of Morita, 10 miles west from the port of Aomori, there were found pieces of micaceous iron-ore and psilomelane, with oval holes pierced from both sides. Apparently these objects were regarded as ornaments or curios, without being applied to any practical use.

Among the traditions given in the *Nihonshoki*, an ancient history of Japan, there is one relating to the mining industry. According to this tradition, in the mythological age Izanami-no-Mikoto gave birth to Kanayamahiko-no-Mikoto, god of mines, and Amaterasu-Okami (the sun-goddess) was armed with swords. It came to pass at one time that she concealed herself in Ama-no-Iwato ("Heavenly Cave"), and several gods made contrivances to entice her out. Among them, Ishikori-dome, a smith, made a mirror with copper obtained from the Ama-no-Kagoyama, by means of a pair of bellows known as the Ama-no-habuki, which was made of a deer-skin. It is recorded also that Susanō-no-Mikoto, her younger brother, observed the abundance of gold and silver in Korea, where his descendants were to reign, and urged them to provide themselves with boats to convey them across the Sea of Japan. In the course of time he accomplished his object, proceeding to Korea with his son, Isotakeru-no-Mikoto. These accounts

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furnish us with a glimpse of the fact that our ancestors in the pre-historic age knew something about the mining industry.

Records for the space of 860 years from the first year of the Emperor Jimmu, 660 B.C., to the conquest of Korea by the Empress Jingu, 200 A.D., show that swords and mirrors were made early in those days, but nothing definite is given as to other metallic products. When the Empress Jingu conquered Shiragi (Sil-la), a province of Korea, the king promised an annual tribute of 80 vessels loaded with gold, silver, silk, and other treasures. This was followed by the surrender of Koma and Kudara (Ko-ku-ryō and Păik-chyöi), two other Korean provinces. The result was a sudden development of communication between Korea and Japan, as a consequence of which scholars, doctors, and makers of various articles, both of utility and artistic merit, made their advent in our country from Korea, and brought the Indian civilization with them. Thus our knowledge regarding the use of metals was considerably improved by Takuso, a smith, who came from Kudara and introduced reforms in the manufacture of metal-wares. In 306 A.D., during the reign of the Emperor Ojin, communications with China supplied the Japanese with a splendid opportunity to bring themselves into a still closer contact with various forms of Indian civilization, which soon spread far and wide through the country. According to tradition, in the reign of the Emperor Kensō, 486 A.D., silver coins were in circulation, which were not, however, the product of Japan. In 552 A.D. Buddhism was introduced from Kudara, Korea. To this religion the Crown Prince Shōtōku became a faithful and enthusiastic convert in 593 A.D., and encouraged its propagation by building numerous temples. As a consequence, there was a large influx of carpenters, sculptors, painters, and artists, creating a demand for metals which was calculated to advance the industry. In 668 A.D., petroleum and asphaltic substances were offered to the court of the Emperor Tenji from the province of Echigo, and in 674 A.D. silver was discovered in the island of Tsushima, to the great joy of the Emperor Tenmu. Silver was offered to the Shinto deities. The governor of the province was raised to the rank of Shōkinge, and honored with several particular favors belonging to that rank, while other officers were rewarded according to their merits. In

691 A.D., the fifth year of the Empress Jitō, the governor of Iyo presented silver and its ores from Miumayama, and Kanzei, a Buddhist priest, made a pigment of white lead.

The people directed their attention seriously to the mining industry, which was specially encouraged under the beneficent reign of Emperor Monmu, full of sagacity and wisdom, from 697 to 706 A.D. During this period the records of mineral discovery stand as follows:

Copper-ore from the provinces of Inaba and Suō; alum from Omi; antimony and its ore from Iyo; cinnabar, realgar, and "shirome" (a speiss or alloy of antimony, arsenic, zinc, bismuth, tin, etc.) from Ise; cinnabar from Hitachi, Bizen, Iyo, Hyūga and Bungo; azurite and copper vitriol from Aki and Nagato; orpiment from Shimotsuke; tin from Tanba, and silver from Kii.

The Emperor sent Itsuse to Tsushima to prospect for gold, while Arakawa was ordered to make researches for gold in the province of Mutsu, and as a result gold was found in the mines of Tsushima, 701 A.D. This pleased the Emperor to such an extent that he gave to that year the title "*Daihō*," signifying "grand wealth," and by way of commemoration of this discovery, Itsuse, the discoverer, was raised in official rank; but alas! to the great disappointment of all concerned, the alleged discoverer turned out to be a fraud. In 699 A.D., the mint, at which copper and silver coins were made, was first established. The zeal of the Emperor for the mining industry was so intense that he took the initiative in the mining-law which is known to us in the famous code, *Daihōryō* issued in June, 701 A.D.

"The people enjoyed the liberty of mining copper and iron in all districts where the government was not itself doing so; but even where this was the case, mining was allowed to taxpayers.

"In case of a discovery of gold, silver, or curious treasures likely to turn out to be useful, information to the government was to be at once given to that effect."

In 708 A.D., the fourth year of the Empress Ganmyō, native copper was presented from the Chichibu county, Musashi. The Empress, elated with this happy augury, went so far as to change the title of the era to "Wadō," which meant "fine

copper." The discoverers, Kusakabe-no-Oi, Tsushima-no-Kataiwa, and Konjōmu, were exalted to the honor identical with that of the governor of a province, while promotion was extended to the officers concerned. The people of this region were released from taxation, while a general amnesty was granted throughout the country. It was during this year that copper money was made in Omi, and in the year following silver coin also was issued. In 710 A.D., copper money was also made at Dazaifu, Chikuzen. This money was known as "Wadōkaihō" (which meant the treasure made in the era of Wadō), but only a small amount of it is now extant.

The striking increase in the demand for copper encouraged its production; and the system of using copper as a ransom for crimes was adopted. In *Yōrōrei*, a code issued in 718 A.D., it was ordained that for capital punishment was to be paid 263 lb. of copper; for the return from exile, from 98.6 to 184 lb.; and so on.

From the geographical records of the provinces, called *Fūdoki*, published in 713 A.D., it was known that Yamato and Mikawa produced mica; Ise, mercury; Sagami, sulphur and alum; Omi, magnetite; Mino, Hida, Wakasa, Idzumo, and Sanuki, alum; Shinano, sulphur; Kōtsuke, azurite; and Mutsu, rock-crystal, mica, and sulphur.

The dissemination of knowledge regarding minerals brought numerous abuses, and at Dazaifu, where the mint was situated, the people were forbidden, in 716 A.D., to have in their houses the alloy "shirome," which facilitated the production of counterfeit coins.

According to tradition, the discovery of the Osaruzawa copper-mine, Rikuchu, was made in 714 A.D. Copper was produced from Ushijima, Kumage county, and Tariyama, Yoshiki county, Suō, in 730 A.D., and being of good quality, it was mined and smelted as the material for the coinage of the mint in Nagato. The mining of copper- and iron-ore was freely granted to the people at large by the *Daihōryō* of 701 A.D., but the mines were gradually assumed by rich and powerful men, as were also the iron-mines in Omi. Thereupon, such monopolization was forbidden, so as to allow the poor to participate in the industry.

The Emperor Shōmu, 724–749 A.D., was a zealous follower

of Buddhism, and built, in Nara, 745 A.D., a large statue of Buddha in a sitting posture, which was 53.5 ft. high, and of which fine copper was required to the amount of 520.5 long tons, and 7,483.2 oz. of gold. For this purpose, the Emperor encouraged the discovery and production of the two metals. The copper was chiefly mined from the Tada mines, Settsu; the Akenobe mine, Tajima; the Naganobori mine, Nagato, and elsewhere in Chōgoku. Placer-gold was discovered by the governor Kyōfuku (descendant of the King of Kudara), at Wakuya, 25 miles NE. of Sendai. He made a present of 392 oz. to the government. Konjōmu, discoverer of native copper in Musashi, was a Korean; while Kyōfufu, Sumusume, and Kojōzan, a smith, who were concerned in the discovery of the placer-gold, were also naturalized Koreans or their descendants. Thus the mining of Japan was greatly indebted to the Koreans.

In 750 A.D. gold was discovered on the coast of Tago, Iohara county, in the province of Suruga. The governor and the discoverer were raised in rank and highly rewarded, and the inhabitants of those districts were exempted from taxes for that year. Several counties north from Taga in Mutsu produced gold, and their taxes were paid in 752 A.D. with 77 oz. of gold. The Tachibana copper-mine, Settsu, became productive, and in Mimasaka magnetic sand was dug from pits. Silver and copper coins were extensively circulated in the year 760 A.D., while in 766 A.D. "shirome" from Hananamiyama, Amada county, Tanba, out of which a mirror was made, was presented to the court, and in 770 A.D. sulphur was mined at Yunotaira, Onuma county, Iwashiro.

As the knowledge of these metals and the crafts utilizing them became disseminated, gold and silver were held in high esteem, as was shown in 646 A.D., in the prohibition of the burial of these metals with corpses. The establishment of the mint at the end of the seventh century increased the demand for silver and copper, and in 815 A.D. the use of gold and silver for ornaments was restricted to the people in certain ranks; while in 834 A.D. the same prohibition was extended to the use of foil and powder of gold and silver through all ranks. The exact amount of the output of gold and silver necessary in those days to meet the demand cannot be gauged. The

provinces of Mutsu and Shimotsuke produced gold, and Tsushima silver, the mine for the latter descending 400 ft. below the surface. In the summer of 864 A.D. it was flooded by heavy rains, and its recovery involved heavy expenses, not supportable by the miners; so that a tax was levied for the purpose of drainage.

According to the *Yengishiki*, published in 927 A.D., the products presented to the government from all provinces were as follows:

Tsushima, 1,168 oz. of silver; Shimotsuke, 192 oz. of placer-gold and 112 oz. of gold-bullion, and Mutsu, 464 oz. of placer-gold. There were also other mines. Records show that in the beginning of the ninth century there were the silver-mines of Ikuno, Tajima; Hosokura, Rikuchu; and Gamō, Inaba. In 902 A.D. gold was abundantly produced from Umegashima, Suruga, while in 970 A.D. Kanase Gorō obtained silver from the Tada mine, Settsu, which was presented to his lord, the famous Tada Mitsunaka. In those days gold and mercury formed articles of trade with the Chinese.

Copper and lead were chiefly produced from such provinces as Nagato, Iyo, Chikuzen, Buzen, and Hizen in the ninth century. The government worked these mines, and built in 818 A.D. a mint for the coinage of copper money at Nagato. At that time the amount of copper minted was 15.54 long tons, and that of lead 7.77 long tons. With the declining production of copper and lead from Nagato, the mint was closed in 868 A.D. But subsequently several mines of Yoshioka, in Bitchu; Okadayama, in Yamashiro; Kafurawariyama and Hichinaiyama, in Mimasaka; Sasayama, in Bizen, and Maruyama, in Iwami, were opened. The government adopted every possible means for the encouragement of the production of the metals either by the impartation of knowledge concerning the art, or by the remission of taxes, or by the supply of miners and smelters. Moreover, the governor of the province was instructed to make a present of a fixed amount of copper or lead; should he make a present of 1.4 long tons of both copper and lead for three years consecutively, he was to be raised in rank. But the supply of these metals was far from meeting the demand. In 940 A.D., the civil war broke out, and the coast of the Setouchi (the Inland) sea was hunted by roving bands of marauders,

by whom the mint of Suō was burnt down. This was followed by the suspension of the official mines.

As previously observed, iron was produced before the eighth century from the provinces of Chūgoku, which were blessed with an abundance of magnetic sand, derived from decomposed granite. In 796 A.D., as the production of iron from Bizen was reduced, the payment of taxes in the shape of iron plows was stopped. It was at the beginning of the ninth century that famous swords made their appearance in these regions. The necessity of military equipments for the provinces on the northern coast of Japan, to guard against threatened invasions, gave birth to famous sword-smiths such as Yasutsuna and his son Sanemori of Ohara (Hoki), who made sharp swords out of the steel produced in these districts.

2. *Second Period.—From 1001 A.D. to 1582 A.D.*

During this period, the production of gold and silver made a gradual but steady increase, particularly in the provinces of Mutsu and Dewa. Fujiwara-no-Kiyohira, the opulent hereditary grand lord of these provinces, built in 1124 A.D. the Chūsonji temple, a splendid building of elaborate workmanship, near Hiraidzumi station on the Tokyō-Aomori route. His grandson Hidehira annually presented 576 oz. of placer-gold to the court. In 1175 A.D., 1,557 oz. of placer-gold from the county of Kesen in Mutsu was presented to Taira-no-Shigemori. After the fall of Yasuhira, son of Hidehira, the large estate, belonging to Minamoto-no-Yoritomo, formed the resources of the Kamakura Shogūnate government. The rich gold-placer was discovered at Shiriuchi, Hokkaidō, during the predominancy of the Shōgun Sanetomo, 1205 A.D. Araki Daigaku, the Lord of Kai, who was despatched with 800 miners to prospect the mines, obtained an immense amount of gold. The placer-gold in Nishimikawa in the Sado province was known about 1000 A.D. The Omori silver-mine, Iwami, discovered in 1310 A.D., produced a large amount of silver from the outcrop.

We may estimate from the following historic records that the industry was extraordinarily prosperous from the eleventh to the thirteenth century. In Chapter II., on the Island of Zipangu, in *The Travels of Marco Polo*, who was the adviser of the Grand Khan Kublai of Yuen, we find the following passage:

"They (the Japanese) have gold in the greatest abundance, its sources being inexhaustible; but the king does not allow of its being exported. Of so great a celebrity was the wealth of this island, that a desire was excited in the breast of the grand Khan Kublai, now reigning, to make a conquest of it, and to annex it to his dominion."

This passage gives the real cause of the invasion by the Tartars, which was brought to an end by their fatal defeat in 1281 A.D. It is curious to note how the tales of Marco Polo at that time excited the brain of a young Genoese and induced him to the discovery of the New World. Later, peace was concluded with the Chinese, with whom Lord Ouchi entered into trade-relations in 1403 A.D. Among the articles of export were gold, copper, and sulphur. These facts, collectively, give a glimpse of the mining industry in those days.

In 1467 AD., the famous civil war of Onin broke out. From that time to the ascendency of Toyotomi Hideyoshi in 1587, the whole country was plunged into the scorching heat of war; nevertheless the mining industry relating to noble metals was not only fortunately undisturbed, but was encouraged by the feudal lords in making provision for their armies. In fact, these mines formed the object of plunder among these warlike tribes. The Omori silver-mine in Iwami producing plenty of native silver about 1530 A.D., offered taxes to Lord Ouchi. The discovery of the Tsurushi silver-mine in Sado, together with the gold-placer washing of Nishimikawa, formed the very coffers of Lord Uyesugi. In 1570 A.D., the Ikuno mine in Tajima produced an immense quantity of silver which proved the rich treasury of Lord Yamana. The Karuizawa silver-mine in Iwashiro, was discovered in 1558 A.D.; its subsequent productiveness resulted in the making of an offer of 17,270 oz. of fine silver to the Lord Gamō. In 1528 A.D., gold was produced from the mines of Suruga, out of which gold coins were minted by Lord Imagawa. About 1540 A.D. the Lord Takeda made 143,930 oz. of gold coin from the mines of Kai and Shinano. These are only parts of the mining industries carried on during the civil war, of which records remain.

The production of copper from the official mines declined in the middle of the tenth century; and the mint being destroyed by the civil war of Tenkei, the coinage of copper money was

suspended, which resulted in the frequent importation from China, by such dignitaries as Shōgun Ashikaga Yoshimasa, of the money necessary for circulation. These facts, however, do not go to prove the decline of the copper-mining industry in those days, since copper was exported by Lord Ouchi, and was also used to make furniture, while descriptions of the prosperity of copper-mines at the beginning of the eleventh century are numerous. For instance, the copper-mines of Nose, in Settsu, and Dōgamaru, in Iwami, became prolific. The mine of Omodani, in Echizen, was discovered. The production of the Yoshioka copper-mine, in Bitchu, was gradually increasing in 1427 A.D.; the Tada mine, in Settsu, produced an immense amount of copper in 1570 A.D.; the Ani copper mine, in Ugo, commenced to be mined in 1575 A.D., and the Motoyama mine, in Mutsu, was also opened about that time. In 1195 A.D., 434.64 long tons of copper were consumed for the purpose of repairing the enormous statue of Buddha in Nara. For the casting of the ponderous bells at Kamakura, in Sagami, and Daigo, in Yamashiro, 1.21 long tons and 13.47 long tons of copper were used respectively. The Emperor Shirakawa, from 1072 to 1128, made as many as a thousand statues of Buddha. All these facts confirm the belief that copper was abundantly produced in those days.

It may be interesting to observe that copper-metallurgy made a wonderful progress. The method of matte-smelting, commonly called the "Mabuki" process or "Yamashitabuki," was invented—which is practiced nowadays at small copper-mines, where the Bessemer process could not profitably be applied, as the cheapest and simplest method of matte-smelting. This method has been improved since the Restoration of 1867, by increasing its capacity and reducing the amount of the consumption of fuel. It was invented in the sixteenth century at the smelter of the village of Yamashita, Tada, in the province of Settsu. Formerly the copper-matte produced from the ore-smelting had been first roasted and then reduced to crude copper with charcoal in the hearth. According to the new process, the matte, without preparatory roasting, was melted in the charcoal fire, and the iron and the sulphur in it were oxidized with a strong blast, as is the case in the modern Bessemer process. The process was conducted in the hearth

in which ore-smelting was carried on. The quantity of matte treated at a time was 658 lb., the charcoal being 30 per cent. of the matte, and the laborers required were a smelter-attendant and bellows-pusher for each smelter, Being a greater economizer of time and labor than the reduction process, it has been gradually adopted in the western part of Japan, to the great improvement of the copper industry.

We have but scanty and meager records of the iron industry during this period. At the Sugatani mine, Iishi county, Idzumo, the smelting-plant for magnetic sand commenced to work in 1266 A.D. The method formerly adopted was a very primitive one, called "Noro" smelting. A hearth was made in the ground, in which magnetic sand was heaped and covered with the fuel. When the latter was kindled the magnetic sand became molten and was reduced, leaving the iron mass in the bottom of the hearth. This method of smelting was crude; but the art of making swords, already prosperous in the ninth century, became more perfect than ever, in order to meet the demands of the time. The swords made by Monju in Mutsu, Masatsune in Bizen, Munechika in Kyoto and Okazaki Masamune in Kamakura, form valuable collections at the present day.

The discovery of the Daira lead-mine in Ugo took place about 1270 A.D., and was followed by that of the Obira mine, Bungo, 1547 A. D. This was started as a tin-mine, but later became a copper-mine. In 1469 A.D., at Inariyama, in Chikugo, a farmer found a coal-seam, which is worked at present as the Miike colliery; while in 1532 A.D., at Kurobeyama in the province of Etchu, sulphur was discovered, and three years later it was worked.

3. *Third Period.—From 1583 A.D. to 1866 A.D.*

Gold and Silver.—During this period, gold- and silver-mining, which had been encouraged in earlier periods, made striking progress; and at the time of Toyotomi Hideyoshi the levels of Hyōtanmabu and Daidokoromabu of the Tada silver-mine, Settsu province, were well known for their prosperity. The former derived its name from the fact that Hideyoshi endowed Hara Tanba and Hara Awaji, the heads of miners, with his family ensign of *Sennari-hyōtan* (Thousands of Gourds), and

allowed it to stand at the entrance of the level in honor of the abundant output of gold. The name "Daidokoro-mabu" (kitchen) was given because all the household expenses of Toyotomi were met by the product from this level. Hideyoshi became very opulent, and divided 21,588 oz. of gold and 154,713 oz. of silver among his numerous lords, 1585 A.D. In 1587 A.D. he made silver coins, and in the year following gold coins of different denominations were minted.

At this time Kanamori Nagachika, the Lord of Hida, ordered his vassal Modzumi Sōtei to make researches for useful ores on his estate, and as a result of such efforts, he discovered, in 1589 A.D., the silver-mines of Kamioka and Modzumi and the Otani gold-mine, from which much gold and silver was obtained later. In 1598 A.D. the discovery of the Shirane gold-mine near the Osaruzawa mine, in Rikuchū, was followed by that of the gold-mines of Komagi and Gojūmai in the vicinity.

It is interesting to observe that in 1573 A.D. Sumitomo Jūsai obtained knowledge of the liquation process from a European who visited Sakai, Idzumi province. After that, gold and silver were extracted by this process from crude copper at the copper-refineries in Osaka. The adoption of the new process contributed a great deal to the increase of the output of silver in our country.

The demise of Hideyoshi was followed by the Regency of Tokugawa Iyeyasu, who adopted the proposal of Okubo Iwami-no-Kami concerning the mining industry. He opened for the first time gold- and silver-mines in the Idzu province, at the same time having a control of the Omori silver-mine, in Iwami. In July, 1601 A.D., the gold- and silver-mine of Aikawa, in Sado, afterwards known as the Sado mine, was discovered. This mine became conspicuous by its large production of gold and silver, and was also brought under the control of Okubo together with the Tsurushi silver-mine and the gold-placer of Nishimikawa. It was during this year that gold and silver currency was coined in abundance. In 1602 A.D., the bonanza of the Waremabu of the Sado mine was prosperous, producing 1,199,327 oz. of gold and silver, which fortune was shared by the Kamayamabu level of the Omori mine, where the silver-yield amounted to 603,946 oz. Several mines in the Idzu province became strikingly productive, the Nawachi silver-mine being particularly conspicuous, since it formed an object

of Iyeyasu's pride. At the Ikuno silver-mine, the Tsukimabu level was productive in 1588 A.D., and was followed by the Sanzensanbyakumai lode about 1600 A.D. The Shirane gold-mine also became productive beginning with the year 1602 A.D., and actually produced 5,277 oz. of gold in 1604 A.D.; 1605 A.D. was blessed by the discovery of the famous Innai silver-mine at Ugo.

Notwithstanding the prosperity of the mines and the rich new discoveries above described, the government did not relax its efforts to develop the mining of these valuable metals. In 1606 A.D., Watanabe Bingo and Sunimokura Genshi were ordered to prospect gold- and silver-mines throughout the country for the encouragement of the production of these metals.

The mining of gold- and silver-ores in those days was only the excavation of the enriched zone above the drainage-level, but as in the course of operations it became necessary to sink below that level, the difficulty of drainage was felt. In 1607 A.D. the productive capacity of the mines of Sado and Idzu became exceedingly diminished, in all probability on account of water, so that 36 mining-experts, selected from Omori mine and the mines of Idzu, were sent to the Sado mine to reform the operation; but in spite of all their efforts, the mine was frequently swamped by water in the rainy season. In the course of his attempt to revive the production of noble metals, Okubo attempted to wash for placer-gold at Shiriuchi, Hokkaido; but the effort proved abortive, owing to the objection raised by Lord Matsumaye Kimihiro, who urged that, since the excessive rigor of the cold climate prevented the cultivation of the land, he was not in a position to find sufficient provisions for the numerous miners. In 1614 A.D., as the Innai silver-mine was troubled by water, the expense for drainage was advanced by the government.

In those days, for the purpose of drainage in the Sasu silver-mine, Tsushima, a treadmill was used. In the Sado mine a well-bucket only was used, which was worked by prisoners convicted of capital offenses. In 1634 A.D., the "Supondoi" was adopted—a primitive hand-pump consisting of a wooden pipe 4 in. square and from 9 to 12 ft. long, having a valve 3 ft. below the upper mouth, and a piston 3 ft. long, with a valve

at its end. In 1637 A.D., Sōho, the hydraulic engineer, who came from Osaka, designed the "Tatsudoi," which took the place of the "Supondoi." It was an Archimedean pump of Roman style, which consisted of a wooden cylinder and screw, 9 ft. long and 1 ft. in the upper and 1.2 ft. in the lower diameter. The ruins of such a pump were lately found at the old working-place, and are now preserved in the Tokyo Imperial University.

The pumps served their turn well in those days. In 1644, the output of the Ikuno silver-mine also declined by reason of the difficulty of drainage. It was at this juncture that Tomomatsu Dōhan mined the levels of Kanaki and Kawato, employing 1,500 persons for pumping, and in the course of two years obtained 35,980 oz. in silver. As the expense of drainage grew enormously heavy in the Sado mine, various taxes were levied from the province in order to meet the deficit. In 1782 A. D., a hand-pump, introduced from Holland, was successfully worked at Aoban and Jingo in the Sado mine, saving £707 per annum from the cost of draining, which in the year following was used to cut an air-way at Seiji level, which had been abandoned on account of bad ventilation.

As the drainage grew to be a serious question for mining, the excavation of the adit was started, utilizing the topographical conditions. In the Sado mine, the adit of Midzukane ravine, whose length was 2,880 ft. through andesite tuff, was commenced in 1629 and completed in 1639. The adit of Ogiri, begun in 1647, was completed after the constant work of 13 years. Its length was 948 ft., and that of the accompanying air-way 1,080 ft. Lastly the adit of Minamizawa ravine was undertaken, which remains to this day. It was commenced July 25, 1691, to drain the water in the working-places at the Waremabu bonanzas. May 14, 1696, success was attained after five years of effort. The length of the adit was 3,014 ft. It was driven from six points, making two intermediate shafts at the Kitazawa ravine. The sections of the adit were made in several forms besides the common rectangular form, in order to reduce the upper pressure and to enlarge the flowing-area, as shown in Fig. 1.

The monthly progress at each working-face was 8.6 ft. through andesite tuff. The excavation was done with chisel

and hammer, making the surface of the wall and the roof very even, and leaving the traces of the chisel. A part where the trace resembles the spider's web, is celebrated as the *Kumonosu-Kengiri.* During the excavation the working was ventilated by brattice. The surveyor of the work was Shidzuno Yoemon, who used the magnetic compass and clinometer, which are now preserved at the Tokyo Imperial University. The circumference of the compass is divided into 480°, and each of these into 10 min. and E. and W. are placed as in the modern compass. The clinometer is 1 ft. square, its scale being divided into 10 *sun* and each *sun* into 10 *bu.* The art of mine-surveying in those days may be inferred from a mine-map, which was made in 1695 A.D., now preserved at the Tokyo Imperial University.

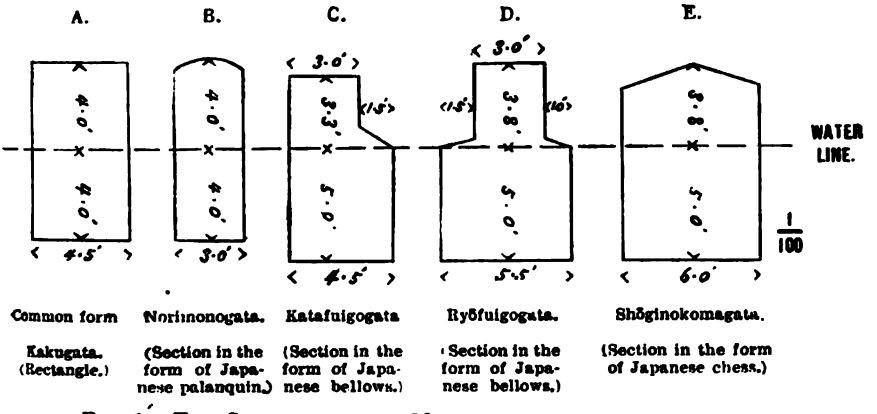


FIG. 1.—THE SECTIONS OF THE MINAZAWA ADIT, THE SADO MINE.

In the Ikuno mine, the adit of Rōmabu was driven in 1608; and in 1638 a drainage-level was excavated from Takeharano for the whole mine. In 1715, the adit Shinkiriyama of the Omori silver-mine was commenced. At the Handa silver-mine an adit was begun in 1741, and its completion in 1746 effected a sudden increase of the output of silver. In 1786, other adits were projected. In the mines of Tada in Settsu, however, owing to topographical conditions, adits could not be effectively used, and the drainage had to be performed by manual labor. Consequently, even in rich bonanza, it was next to impossible to work below certain horizons, and the work had to be abandoned.

In ancient times, much difficulty was experienced in the transportation of supplies and products. As the metal-mines were always situated in mountainous regions, the provisions for miners had to be brought from distant places; and in seasons of famine, it was a serious question how to keep a vast number of working-men from starvation. In the famine of 1642 the Karuizawa silver-mine (Iwashiro) had to be abandoned, but the Ikuno mine escaped a similar fate by working the bonanza of Sanzensanbyakumai lode. In the famine of 1643 the Sado mine was a great sufferer.

Through the efforts made by Okubo Iwamino-kami for the production of the valuable metals, the mining industry made a rapid and healthy growth. In the Ikuno mine, levels turned out to be very prosperous in 1622, yielding a rich ore (0.4 to 0.6 per cent. Ag), and the monthly output of silver was 21,588 oz. The total product of gold in Japan in 1630 was 23,061 oz., of which 11,889 was from Sado, 9,454 from Satsuma, and 1,420 from Idzu. In 1634, the Nobezawa silver-mine, Uzen, became productive, and was converted into a government undertaking, and in those days, there were 53 productive levels. The production of 1635 from the Ikuno mine was 143,917 oz. of silver. The province of Satsuma was known from ancient times as the chief gold producer. In 1638 the discovery of the famous Yamagano gold-mine was made, producing 31,380 oz. of gold annually; and another discovery occurred in the well-known Serigano gold-mine in 1652. In 1664, the decline of the Sado mine became decisive. It yielded only 117,509 oz. of silver, owing to the water trouble at the Waremabu bonanza; and it was finally abandoned. Though the Ikuno mine had been thus impoverished in 1644, it gradually regained its capacity after 1668, and in 1671 it was producing 119,932 oz. of silver per annum. In 1669, the ore of the Osaruzawa gold-mine was changed into copper-ore, and it is now being worked as a copper-mine. In 1674, the working Suwamabu, in the Waremabu bonanza of the Sado mine, furnished ore containing 0.29 per cent. of silver and produced 5,037 oz. of silver in 10 days. The annual output of silver of that year is given as 239,599 oz. The Ikuno mine declined after the prosperity of 1671, but was fortunately checked by the flourishing condition of the Ryōgokuyama

level. Thus in 1683 it produced 42,785 oz. of silver, and in 1696 the output amounted to 111,092 oz. of silver.

In the sixteenth century, the European ships visited the western shore of Kyūshū, at Hirado, Gotō, Hizen, and Satsuma, and entered into commerce with the natives. The amount of gold and silver then exported by the Portuguese was enormous. In 1607, the Dutch ships were granted permission to trade at any ports in Japan. After the Christian massacre at Shimabara, 1638, the trade was confined solely to the Dutch, Chinese, and Koreans at Nagasaki. The decreased production of silver in Japan raised its value exceedingly, so that in November, 1662, the value of 1 oz. of gold was equal to that of 12 oz. of silver. In order to decrease the amount of silver exported, the gold exportation was granted in 1664, and the rate of exchange was 1 oz. of gold equal to 14 oz. of silver. The amount of silver and gold exported by the Dutch and Chinese vessels during the 20 years, 1648 to 1667, was 33,222,559 oz. of silver and 55,472 oz. of gold. The outflow of silver was not decreased, so that the exportation of gold and silver was summarily prohibited in 1668. In 1672, in trading with the Dutch, the use of gold and silver was permitted at the request of the latter, on the ground that unless silver were brought to their country from Japan, silver would be highly appreciated in Holland, causing great difficulty in business. In 1685 a limit was put upon the total amount of trade to be done, guarding against the reckless export of gold and silver; but the foreigners invited the merchants of Japan to private trading on the sea. Thus the exportation of gold and silver was not altogether stopped. The total amount of precious metals exported during 164 years, 1601 to 1764, was 3,763,572 oz. of gold and 135,768,918 oz. of silver. With the diminution of gold and silver currency, commerce in Japan was much disturbed, and the limited production of those metals from our mines was far from satisfying the demand. Hence, to increase the amount of currency, the recasting of the old coins, lowering their fineness, was commenced in 1695. This proved to be a failure, as may be seen in the rise of the price of commodities. Under the circumstances, the increase and improvement of coins formed a subject for serious debate in those days. In August, 1695, it was declared that, if one found mines of gold, silver,

or copper, he was at perfect liberty to mine the same at his discretion.

In 1697 they began to use, in the Sado mine, stamps driven by water-wheels to crush the poor ores formerly abandoned. The annual output of that mine in 1703 was 18,788 oz. of gold and 667,636 oz. of silver. The Ikuno mine was prosperous in 1709, yielding 123,495 oz. of silver. The total output of silver in Japan in 1711 was 479,731 oz., and that of gold, 7,533 oz. In 1717 the Sado mine was greatly impoverished, producing only 131,802 oz. of silver. In 1726 rich ores were produced from the Shinkiriyama adit of the Omori mine, and the output of that year was figured at 64,053 oz. of silver. In 1767 the Ikuno mine produced 82,920 oz. of silver.

In 1771 the Sado mine produced 5,177 oz. of gold and 621,806 oz. of silver. It is very curious to observe that the methods of rugging and concentrating the ore at that mine were identical with those of Cornwall in ancient times, as may be seen from the following description: The ore was at first brought into the rugging-room, *kanaba*, where a slab of andesite was prepared on a base which was 3 ft. high and 3 ft. wide, at the front of the window. There it was rugged into slime with a hammer, and then was sieved with a fine screen. The slime was introduced into *tateoke*, a dolly-tub, with water. After stirring with a ladle, the side of the tub was beaten to settle the deposit. The concentrates were subjected to vanning with a plate vanner. The headings were made into balls and roasted on a charcoal-fire. Then the roast was smelted with lead, 50 per cent., and iron, 5 per cent. The lead produced was cupelled; the matte was also smelted to absorb silver into the copper; the latter was liquated; and then the lead was also cupelled. The auriferous silver was smelted with sulphur, producing silver-matte and silver rich in gold. From the former the sulphur was driven, and the silver was absorbed in lead, which was then cupelled. The latter was crushed fine, then piled in a conical form on a porcelain basin, mixed with common salt, and roasted. The roast was washed to separate the silver chloride from the gold. The silver chloride was filtered and cupelled. The silver was considered very fine at that time, but it contained 0.61 per cent. of gold. The gold-bullion produced was 995.4 fine.

Tanuma Mototomo, a favorite of Shōgun Iyeharu, undertook to prospect for gold at the Mt. Kimbusen in Yamato and Chichibu in Musashi. The former place was reported as rich in ancient times. After unsuccessful working for several years, the work was stopped in 1786, when the Shōgun died.

In 1804, the Aoban and Torigoye stopes of the Sado mine were impoverished, reducing the output to 578 oz. of gold and 6,623 oz. of silver. The Yamagano mine, Satsuma, also declined in 1805, producing only 1,775 oz. of gold annually. The Ikuno mine produced only 33,821 oz. of silver that year, but gradually recovered, and produced 85,152 oz. in 1812. In 1814, the Omori mine became prosperous, yielding annually 27,618 oz. of silver. The Innai mine was productive, yielding annually 57,251 oz. of silver from 1741 to 1817. In 1823, the Sado mine made some profit after 28 years of loss; but its production did not exceed 1,186 oz. of gold and 30,147 oz. of silver. In 1840 the production of the Handa mine was increased by draining the water in the Okutate and by the opening of the Nikaimabu. Its annual output, for 146 years from 1719 A.D., was 17,037 oz. of silver. In 1861, the Kosaka silver-mine, Rikuchu, which is nowadays the champion copper-mine in Japan, was discovered. After five years Lord Nanbu built a shaft-furnace and a cupelling-hearth to develop this mine; but it was disturbed by the civil war of 1867.

Copper.—At the end of the sixteenth century the Kawakami mine, in Harima, and the Hitachi mine, in Hitachi, were discovered, and the Maruyama mine, in Iwami, was prosperous, which was followed by the discovery of the Gamō mine at Iwashiro. In 1610, the Ashio mine, in Shimotsuke, which is now the largest mine in Japan, was discovered. The Yoshioka mine, Bitchū, was prosperous at that time, and was converted to be a government mine in 1642, yielding 30 long tons annually. The Ashio mine was also developed and became a government mine in 1647. In 1669 the Osaruzawa gold-mine was changed into a copper-mine as mentioned above. In 1670 the Ani copper-mine was discovered. Thus the copper-mining industry made a slow, but steady, progress.

In copper-mines, as in those of the precious metals, the working is easy when the excavation relates to the upper part near by the outcrop, but the difficulty caused by water arises

when depth is reached. Hence in the middle of the seventeenth century the Naganobori mine, in Nagato, and Maruyama mine, in Iwami, which were productive, had to be suspended on account of difficulty of drainage. The Yoshioka mine was brought into working order only by the excavation of several adits.

In the fifteenth century copper was exported by Lord Ouchi to China, and this trade continued to thrive in the time of the Tokugawa Shogunate. In 1638 Sumitomo Rihei and 21 merchants were granted permission to trade copper with foreigners, whereby the amount of export was greatly increased. But in 1668 the export of copper was prohibited, together with that of gold and silver. Sumitomo Kichizaemon enjoyed the special privilege of exporting copper, since his ancestors had been engaged in the trade since 1573. At the time under consideration there were 23 copper-mines in working order, which after the lapse of 17 years, were increased to 34. Of these the Ashio mine stands conspicuous for its prosperity, as it produces 1,488 long tons per annum, which is followed by the Yoshioka mine, producing 11.75 long tons annually. The total annual production of copper at that time was 5,357 long tons, from which 13,648 oz. of silver was liquated. One-ninth of the copper was consumed in Japan, while the rest was exported. The laborers in copper-mines numbered 200,000, and those engaged in liquation at the copper-refineries at Osaka, 10,000. In 1688, as the Ashio mine was declining, the capital for recovering was advanced by the government, but the recovering process involved a great amount of difficulty. In 1690 the engineers of Sumitomo Kichizaemon, who was working the Yoshioka mine, discovered the Besshi copper-mine, Iyo, one of the largest mines in Japan, in which mining was commenced in the following year under the government license As the deposit was a very extensive cupriferous pyrite bed in the crystalline schist, it produced 178.5 long tons of copper in the first year, and in 1695 595.24 long tons annually. The year 1700 was also signalized by the discovery of the Arakawa mine in the province of Ugo. In August, 1695, the government urged the opening of the copper-mines with the mines of gold and silver, to increase their production to satisfy the demand.

In 1689 the amount of trade was fixed, so that the Dutch used to have a certain amount of commodities left untraded. In 1695 A.D., Fushimiya, a merchant in Nagasaki, applied to the government for permission to purchase these goods with copper, within a specified limit, which was granted. Thereby the export of copper was raised to 5,298.8 long tons in 1697, while, at home, the supply did not meet the demand. In 1702, the government summoned Sumitomo Kichizaemon to Yedo to undertake the increase of the production of the mines of Besshi and Yoshioka, which he was working. Acting under the suggestions of this experienced miner, the government advanced 5,000 *ryō* as a mining capital for the Besshi mine, and the same amount for the excavation of an adit for the Yoshioka mine; and as the former was located in a mountainous region which was handicapped in communication-facilities, 30,914 bushels of rice was granted by the authorities to purchase at nearly one-half of the market-price; and the products from these mines were extensively exported. It was in this wise that the so-called "government copper-mines" were brought into existence, of which the government required a certain fixed amount to be produced for specific objects, leaving any surplus to the disposal of the miners.

In 1711, the total output of copper in Japan was 3,809.52 long tons, of which 2,857 long tons was exported. The amount of the exportation fell in 1715 to 2,628.60 long tons, of which 1,785.74 long tons was exported to China and 843.60 long tons to Holland. In 1728 the exported copper amounted to only 758.83 long tons, that is to say, 245.54 long tons from the mines of Akita, 60.36 long tons from those of Nanbu, 395.33 long tons from the Besshi mine and 53.60 long tons from the Yoshioka mine. In March, 1736, the government urged the starting of the mines of copper, lead, or tin, with the purpose of increasing the production, and promised various facilities, as a consequence of which the Kune copper-mine (Tōtōmi) and the Kusakura copper-mine (Echigo) were discovered.

Copper for exportation was collected by the merchants at Nagasaki, but the amount never equaled the demand. After 1738, copper was collected at the Osaka Copper Office and sent

to Nagasaki, thus affording every possible facility. In 1754 the amount of copper supplied from the government copper-mines for export was 1,586.01 long tons, of which 734.27 long tons were contributed from the mines of Akita, 428.81 long tons from those of Nanbu, and 422.94 long tons from the Besshi mine. But later the amount was necessarily decreased by reason of the diminished productive capacity of the Akita mines.

The price of copper purchased by the government from the government copper-mines was 361 oz. of silver per long ton of copper, as in the Besshi mine. In 1750, it was reduced to 279 oz. of silver, against the market-price of 420 oz. Hence Sumitomo, the owner of the mine, petitioned frequently to raise the purchase-price, without avail. In 1767, the government price was raised to 305 oz. of silver; but, as the production of copper was scanty, the market-quotation was much higher. Consequently, such an honor of title as to be called "the government mines" was regarded as rather a sort of misplaced kindness to the owners of the mines; but it appears that in spite of the contrary statement made by them, the government's protection certainly went a long way to help them. The price of copper for the Chinese trade was 231 oz. of silver per long ton of copper, and that for the Dutch 121 oz. Hence the copper trade of those times involved heavy losses. Moreover, the "Kakeire Dō" (or surplus copper to fill up the losses of transportation) was added at the rate of 1 per cent. for Dutch export and one-third of one per cent. for Chinese.

The Ashio mine, which had been included in the list of the government mines, was excluded by its decline after 1736. For five years, beginning with 1742, the mine was authorized to cast a copper money known as "The Kanyeitsūhō," which was specified with a letter, Ashi, from the name of mine, on the back surface. In 1747, the annual output of copper amounted to 110.14 tons. The government spared no energy to bring about an increase of production; and in 1763, instructions were issued to the effect that copper-mines, which had been suspended or left untried, should be thoroughly investigated, and that the government should be informed of the results. In 1775, the government itself started to prospect copper-ore at the Mt. Kurama, near Kyōto. By such hard

work the production of copper was increased, while the demand for export was satisfactorily met.

At the end of the eighteenth century, the Daikoku lode of the Omodani copper-mine, in Echizen, was discovered, which offered to the government 73.8 long tons of copper. It was about this time that the excavation of an adit at the Sasagatani copper-mine, in Iwami, was started, but not meeting with ore-deposits, the attempt ended in failure. In 1821 the Ashio mine had finally to be abandoned; but the Besshi mine was prosperous, yielding from 400 to 600 long tons of copper per annum; and about 1840, the Tenwa copper-mine, in Yamato, and in 1865 the Hōmanzan copper-mine, in Idzumo, were discovered, and almost simultaneously a wide vein was found at the Gamō copper-mine, in Inaba, which proved to be quite a prosperous one. In 1860, the export copper supplied from the government copper-mines was 1,086.71 long tons, of which 352.44 tons was produced from the mines of Akita, 311.32 tons from those of Nanbu and 422.93 tons from the Besshi mine; the mines of Tada, in Settsu, were productive for 189 years from 1662 to 1868, producing 7,152.40 tons of copper—with the exception of 18 years in the interval during which the work was suspended.

For 85 years, from 1755 to 1839, the copper exported by Dutch vessels was 39,042.80 long tons, and that by Chinese vessels was 61,411.87 long tons. For 257 years, from the first year of the inauguration of the Tokugawa Regency, 1601, to 1857, the total amount of the exported copper was reported to be 319,922.15 long tons; that is, the annual export was 1,244.85 long tons.

The excavation of rocks and ores was formerly effected by means of chisels and hammers, but where they were hard, fire-setting was sometimes used, as at the Nawachi silver-mine, in Idzu, and in the Besshi copper-mine, to facilitate the mining of the ore. The excavation by such means must have been slow and tedious. In 1862, Raphael Pumpelly and W. P. Blake, both American engineers, and Oshima Takatō, introduced for the first time blasting with gunpowder at the Yūrapu lead-mine, in Hokkaido, which was considered as a forerunner of the new working of the mining industry during the present *régime*.

Iron.—Iron was formerly smelted from the magnetic sand produced from decomposed granite which occupies the mountainous region of Idzumo, Iwami, Bitchū, Bingo, and Aki. Owing to the progress of society and the consequent increase of the demand, the iron industry has grown to be quite prosperous. In the valley of the Tōjō river, Bitchū, places worked for magnetic sand numbered 267 in 1671, from which the inference may be drawn of the remarkable growth of the business, which must be attributed to the great protection rendered by the feudal lords. It is quite an opportune time to describe the remarkable invention of balance bellows in Iwami at the beginning of the eighteenth century.

In the iron metallurgy, the tread-bellows, "Fumidatara," was formerly used, which required eight men to operate. The new balance bellows, "Tenbindatara," which was invented in the eighteenth century, needed only two men for the pair to produce the same effect. This saving of smelting-expense was conducive to the remarkable development of the industry. As a consequence of this growth a conflict of interests with farmers began to arise. For instance, in 1846, the people in the valley of the Takahashi river (Bitchū) lodged complaints with the government to stop the washing of magnetic sand on the upper course, because the water of the river became thereby so turbid that it was utterly unfitted for irrigation, and the river-bed was raised high by sand deposits. Through the arbitration of Lord Asano, the matter was brought to a successful issue on the ground that the washing was executed at the time when agriculture was inactive. Another industry was started in Tosa in 1782, and the magnetic sand on the coast of the Hokkaidō became the object of attention. In 1801, permission was asked for the smelting of iron in the environs of Hokodate, but was not granted by the government. In 1855 Takenouchi Yasunori and Takeda Hisaburō built shaft-furnaces at Kobui, Oshima, Hokkaidō, to smelt magnetic sand from the coast. This was the first one built in Japan. The Kamaishi iron-mine, Rikuchū, was discovered in 1823, where the smelting was commenced in 1849, and in 1860 shaft-furnaces were built by Oshima Takatō to smelt the iron-ores. In 1857, the Kamiteoka (or Kamichūka) iron-mine (Iwaki) was discovered, where tread-bellows were used in smelting ores.

Other Metals.—With reference to other metals, the want of records prevents a complete description. Lead was chiefly produced from the silver-mines; the output of this metal in 1710 was 14 long tons. Tin was mainly produced from the Taniyama mine, in Satsuma: the output being 76.36 long tons in 1849, but in 1859 it was reduced to 17.62 long tons. Antimony was produced from the Ichinokawa mine, Iyo, 1736 A.D., and used for the preparation of drugs and alloys.

Coal.—In the counties of Onga, Kurate, and Kaho, in Chikuzen, which are the important coal-fields in Japan at present, coal has been produced since 1702 in a somewhat limited degree. In 1721 the coal at Hiranoyama, in Miike coal-field, in Chikugo, began to be worked. This was followed by the discovery of the coal-seam of the Yoshinotani colliery; and in 1800 Goheida, a native of Hirado, discovered the Takashima colliery in the province of Hizen—and in these districts coal was known as "Goheida," after the name of the discoverer. In those early days, coal was chiefly used for domestic purposes, but sometimes it was sold as a fuel in the salt-fields in Setouchi coast or to foreign steamers visiting Nagasaki. In 1843, Matsumoto Heinai started a coal-store at the port of Ashiya, in Chikuzen, where he monopolized the business of the coal, eggs, and crude wax produced in that province. In 1855, a steamship was sent as a souvenir to the Shōgun by the King of Holland. This suspended the export of block coal from Chikuzen, since it had to be used as fuel for the steamer. The maximum output of coal from the province of Chikuzen was fixed by Lord Kuroda at 60,000 long tons annually. Coal-mining in those days was simply cutting from the outcrop, and hence there was every need for improvement. In 1855, the Oura slope of the Miike colliery (Chikugo) was started. It reached success in four years, when the coal-seam was met at 230 ft. on the slope, or 40 ft. vertically. The Takashima colliery introduced in 1867 the European method of mining.

Petroleum.—The petroleum in Echigo was formerly obtained from ditches excavated along the petroleum-seepage, as seen at Kurokawa. This was, however, changed in 1818 to the pit-

sinking method. The utilization of the natural gas from the oil-fields of Garameki village, Kanbara county (Echigo), was projected early in 1613, attracting but little attention on the part of the people.

At the end of the Tokugawa regency the country was in a troubled condition, whence the mining industry was also disturbed, and at last stopped during the civil war ending with the Restoration of 1867, which has given a happy chance for mining development.

From the earliest times on record several persons have devoted themselves to the development of the mining industry in Japan. Especially is Modzumi Sōtei, in Hida, worthy of mention for his great efforts in this cause. Naturally, after his death, he was revered and deified by the miners in that region. At the beginning of the Tokugawa regency, Okubo-Iwaminokami-Nagayasu exerted himself, as described above, to work the gold- and silver-mines, so as to build up the natural resources. When the trade was started with foreigners, an enormous amount of gold and silver was taken from Japan, which reduced the domestic supply to such an extent that the scarcity of coins in circulation was keenly felt, while the export of copper was no less large; so that at the end of the seventeenth century the development of the mining industry was the most serious question of the time. At this juncture, Japan was blessed by the birth of a hero at the northeastern part of Honshū, whose name was Satō Nobukage. He was born at Nishimaonnai, Okatsu county, in Ugo, in 1673 A.D., and acquainted himself with the science of administration and agriculture after his father Nobuhide. He opened the Matsuoka silver-mine (Ugo), and worked it, as he has explained in the *Kōjōhōritsu* (*The Administration of Mines*), 1688–1703. Later, he mined the tin-mine at Ashio, Shimotsuke, 1704–1710, and the tin-mine at Takeda, Bungo, 1716–1731. He is the author of *Sansōhiroku* (*The Principles of the Exploitation of Mines*), which contains epitomized accounts of his rich experience. In August, 1731, at the invitation of one of his pupils. he paid

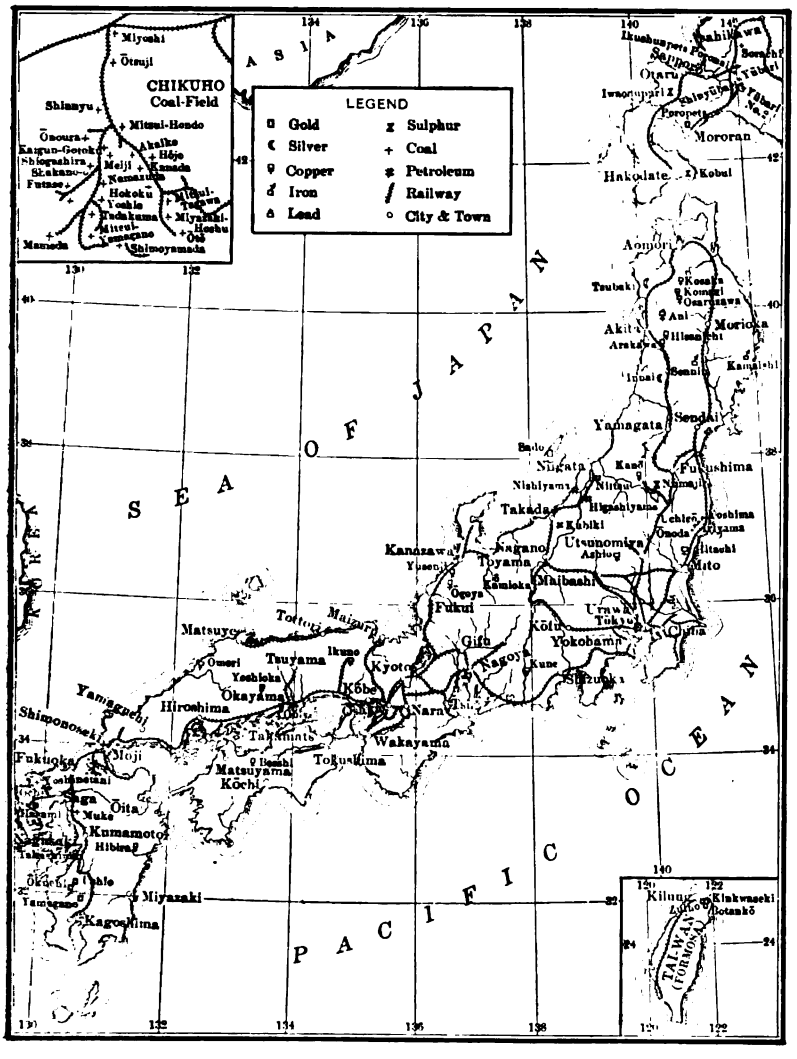


FIG. 2.—MAP OF JAPAN, SHOWING IMPORTANT MINES.

A. The Dōyū Outcrop.
FIG. 3.—THE SADO GOLD-MINE.

A. The Daikinkwa Outcrop.
B. The Mayekinkwa Outcrop.
FIG. 4.—THE KINKWASEKI GOLD-MINE, FORMOSA.

FIG. 5.—THE KOSAKA COPPER-MINE.

FIG. 6.—SMELTING-PLANT IN SHISAKA ISLAND, WHICH BELONGS TO THE BESSHI COPPER-MINE.

FIG. 7.—THE ASHIO COPPER-MINE.

FIG. 8.—THE IMPERIAL STEEL-WORKS, KYUSHU.

FIG. 9.—THE MANDA SHAFT, MIIKE COLLIERY.

FIG. 10.—THE NISHIYAMA OIL-FIELD, ECHIGO.

a visit to the Ani copper-mine in Ugo, where, in the course of his examination underground, an explosion of gas ended his glorious and promising career.

There are even to these days many miners in the provinces of Mutsu, Dewa, Iyo, Tajima, and Iwami who have full confidence in his principles of the exploration of mines. His son Nobusuye was quite equal to his father in point of scholarly attainments, and in 1781 he prospected the gold-mine at Shinjo, Uzen, and then he visited the Ashio copper-mine, where he taught a liquation process. In 1784, while he was devoting himself to reopen the tin-mine which his father had owned, he was affected by poison from the arsenious acid, which proved fatal. His son, Satō Nobuhiro, was educated by his father and by celebrated scholars in Yedo in those days, and, accepting the request of his pupils, he examined the Ani copper-mine. Since his scholarly attainments were far in advance of the requirements of the time, he had to retire to Kadzusa from public life, in order to avert the suspicion of the government. Then he devoted himself to the reform of household science, and wrote 60 volumes on agricultural administration, besides revising his grandfather Nobukage's *Sansōhiroku.* He reduced to writing *Kōjōhōritsu,* which was inherited from his grandfather by oral teaching, as his son was too young to understand and remember it. He passed away Jan. 6, 1850.

II.—After the Restoration.

1. *Before the Japan–China War* (*from* 1867 *to* 1893).—Before the Restoration of 1867, the mining and metallurgical arts in Japan had reached the highest development which could be attained without the aid of mechanical contrivances—the only source of power available in that period having been manual labor. The condition was identical with that of Europe, 300 years ago. If in those days a further progress had been desired, it would have been turned towards a new direction, based on systematic science and refined economy.

The Restoration wrought in every direction throughout Japan, the greatest change in accordance with the requirement of the times. It was no wonder that the new government took every opportunity to encourage the mining industry. As the commencement of this policy, an act was issued in February, 1872, by which the Bureau of Mines was established in Osaka, which was formerly called the Osaka Copper-office, and which became the embryo of the present Bureau of Mines, Tokyo. A mining-law, *Nippon Kōhō*, was issued in January, 1873, according to which, whoever wanted to work a mine, had first to make application to the local office. If the people in that province did not lodge any complaint, the central government issued instructions to the local government to grant permission to work the mine during a certain term.

In order to lead the mining industry into a new trend, fresh sources of knowledge were sought in the Occident. At that time, the government took some of the larger mines under its control, and also supplied to them the knowledge and capital which they required. For this purpose, the Ikunos silver- and copper-mine, in December, 1868, and then the Sado gold-mine, the Kosaka silver-mine, the collieries of Takashima and Miike, the Okudzu gold-mine, the iron-mines of Kamaishi and Nakakosaka, the Innai silver-mine, and the Ani copper-mine—ten in all—were selected as the government mines successively. In 1870 the Bureau of Mines engaged Baron von Richthofen, a German geologist, to prospect mines and to promote the establishment of a mining-school. Our government utilized most wisely the services of mining engineers, civil engineers, geologists, and other instructors, and even of miners

from England, America, France, and Germany, numbering in all 78 men, for the reformation of various industries in Japan. The foreign engineers were distributed to the government mines to direct the working, and in order to prosecute their designs, the government invested the required capital, and thus gave practical examples of mining, which aroused the people from their long sleep in this industry. Not only were the mines supplied with capital, but also the people learned the foreign arts, which were finally propagated in all the mines throughout Japan. Among the eminent mining engineers now engaged in the business, there are not a few who were directly educated by these Occidental engineers.

The government mines having undergone much improvement in consequence of the policy above mentioned, the government afterwards thought it high time to hand over the mines to private enterprise. Accordingly, in 1885, all the government mines, except the Miike colliery, the Ikuno, and the Sado mines, were disposed of to the people. At the same time the engineers who had been educated by the foreign engineers, took the place of the latter in working the mines. Afterwards the Miike colliery was sold to a private operator in 1888 A.D., and the Sado and Ikuno mines were assigned to the Bureau of the Imperial Estate in the following year, leaving the Shinbaru and Gotoku collieries, worked for the navy, as the only mines under government control. In short, the unremitting efforts of the government in the industrial sphere have been modified from time to time, as the conditions required.

The progress of society did not allow the continuation of such an arbitrary system as the mining-law, *Nippon Kōhō*, already mentioned, which was calculated to impair seriously the advance of the mining industry. In 1890 the law was amended; and the new regulation, *Kōgyō Jōrei*, was issued two years later. The concession system, which distinctly established the right of permanent working, introduced a sound development of the mining industry in Japan.

The government did not forget the necessity of technical education. In September, 1870, it laid a base of mining education by the erection of the Mining School, *Kōgakuryō*, at the instance of Oshima Takatō. Afterwards it was changed to the Engineering School, to teach not only mining, but all other

branches of engineering, by foreign instructors. In January, 1877, the courses of the school were raised, and the name was changed to the Engineering College. In September of that year other courses of lectures on mining and metallurgy were commenced in the Science Department, Tokyo College. Both were included in 1886 in the Tokyo Imperial University.

The Progress of Mining Practice.

Gunpowder was used for blasting in the Yūrrapu mine, Hokkaidō, in 1862, as before mentioned, and the practice was immediately followed by the Sado mine discontinuing the old means of drilling and fire-setting, and in the year of the Restoration, 1867, the Ikuno mine adopted also the blasting method. During the 18 years between 1868 and 1885 there was much activity in the introduction of new Occidental technical knowledge in several departments of mining. The laying of rails both underground and overground, the sinking of shafts and the driving of levels, the introduction of stoping systems (including the pillar and long-wall systems for coal), methods of timbering, the deep boring for exploration, the application of horse-, water-, and steam-power, the use of winding-engines, pumps, and ventilators, ore-breakers, dressing-apparatus, and reverberatory furnaces, the methods of amalgamation and lixiviation of gold-, silver-, and copper-ores, the methods of assaying, and the method of underground surveying and mapping—are the principal items of improvements introduced from foreign countries.

From 1885 to 1890, rock-drills, dynamite, and high explosives came into extensive use, while effective up-to-date machines were adopted in rapid succession—such as aerial rope-tramways, Huntington mills, Frue vanners, and other new dressing-machines, together with Piltz furnaces, water-jacket furnaces, steam boilers of new types, water-turbines, electric deposition, etc. The adoption of electricity generated by water-power at the Ashio copper-mine, in 1889, called the attention of mining-operators to the great technical and economical advantages of this agency, and greatly accelerated the progress of the mining practice. In 1893 the Bessemer process for copper was introduced, which has produced a striking effect on the copper industry.

The iron-metallurgy of Japan has very ancient records to

show. In the districts of Chūgoku, pig- and wrought-iron and steel were prepared from magnetic sand to the satisfaction of all the domestic demands. In 1874 the government started iron-smelting works after the most up-to-date method in the Kamaishi iron-mine; but the enterprise proved, unfortunately, to be a failure. Ever since that time charcoal pig-iron has been made in Kamaishi, Hitokabe, and the environs of Morioka city. The iron necessary to supply the deficiency of the domestic production has been imported from abroad.

In the coal-fields of Kyūshū, which furnish at present 65 per cent. of the total output of coal in Japan, the Shakano-o colliery led others into the setting up of boilers in 1881. In Hokkaidō, coal was first worked in 1883, under Benjamin Smith Lyman, an American geologist. During the period under consideration, the domestic consumption of coal was quite insignificant. Railways were laid in 1880 between the collieries in Hokkaidō and the harbor of Otaru, and in 1891 the mining-railways of Chikuhō introduced a new feature in coal-mining. After the introduction of American rope-drilling into the Amaze oil field in Echigo, the oil industry achieved positive economical successes.

2. *After the Japan-China War* (*from* 1894 *to* 1908 A.D.).—In 1894 arose the Japan-China war, which ended in the victory of Japan. Subsequently, with the sudden uprising of various branches of industry, a new foundation of mining activity was soundly laid.

In 1897, the cyanide process was introduced into the gold-mines in Satsuma, and this method has rapidly been propagated throughout all the gold-mines of Japan. Mechanical roasting was introduced at the Ashio copper-mine in 1900; and in the same year the Kosaka mine, adapting pyritic smelting of its ores, took its place as a leading producer of copper, gold, and silver, instead of being only the largest silver-mine. These steps have practically revolutionized copper-ore smelting in Japan. Copper blast-furnaces have been gradually enlarged. For instance, the furnace at the Kosaka mine, 60 by 3.3 ft. in size, was the largest in the world when it was erected in 1907, and afterwards (June, 1908), in the Kano copper-mine, a furnace 65.5 by 3.3 ft. in the section at the level of the tuyeres was built. The most noteworthy recent events in the mining world are

the open working of the massive ore-deposits in the Kosaka copper-mine; the utilization of zinc-blende, by means of magnetic separation, in the Kamioka and the Kanō mines; the adoption of lime-roasting in the copper-mines of Hitachi, Ashio, Kamioka, Osaruzawa, Kosaka, Besshi, etc., and, finally, the practice of the flotation process for zinc-ores in the Kamioka mine.

Cupriferous pyrites have begun to be used as sulphur-ore, the residue being utilized as flux in the smelting of acidic copper-ores. The demand for the pyritic ores grew larger with the increased use of phosphate fertilizers, in or about 1900; custom-smelters on a small scale appeared after 1890, and 12 custom-smelters were once worked in and about the Inland Sea, when the sulphuric acid and the phosphate fertilizer manufacturing industry became more prosperous than ever. In fact, the mines of Kosaka, Hitachi, Ikuno, Yoshioka, and others have actually begun to purchase ores. Thus it will be seen that the metallic mineral industry has been gradually and vigorously developed upon an independent foundation.

After the Japan-China war, popular opinion demanded the establishment of iron-works by the government. To meet this necessity, the Imperial Steel Works, in Kyūshū, were opened in 1901. Their capacity was intended to be one-half of the annual demand, which was estimated to be 120,000 long tons. Since then, the plant has been gradually enlarged to 150,000 long tons at present; one blast-furnace has been erected at the Sennin iron-mine, for the manufacture of charcoal-pig; and at the Kamaishi mine, the plant has been enlarged, and Siemens-Martin furnaces have been built. In 1909, at the harbor of Mororan, an experimental blast-furnace was blown in. Another steel-foundry is now being built in the same place, under the co-operative efforts of the Hokkaidō Colliery & Steamship Co. and the two firms of Armstrong and Vickers & Sons.

In 1898 coal was first exported to a certain place on the Chinese coast. The collieries of Kyūshū were gradually deepened, and many of the slopes in the district attained the depth of several thousand feet. The extra cost of timbering, transportation, and drainage, rendered slopes uneconomical, and at the beginning of 1900, a shaft 700 ft. deep was completed in the Shin-nyu colliery, and in 1902, the famous Manda No. 1 Shaft,

896 ft. deep, was finished in the Miike colliery. New shafts 1,000 ft. deep are being sunk successively in the collieries of Ita and Futase. Those collieries have gradually adopted electric power, which took the place of steam-boilers in use. Since "special" pumps were first introduced in the colliery of Kyūshū, they have been adopted in general except in the Miike. Though these pumps are uneconomical, yet they satisfied their demand, because water is found in but a small quantity in all collieries in this region. However, since 1900, other economical pumps have been adopted, such as the Evans and other electric pumps. As regards the improvement of the pumping-apparatus, the Miike was always a leader, affording great benefits to other collieries, because it requires a supply of great pumping-power (about 25,000 h-p.) for regular operations and for experiments. Together with this technical progress, the amalgamation of mining concessions and the enlargement of working-plants have also induced the rapid advancement and the success of the coal industry since 1900.

The progress of mining industry called for the reformation of the mining-law, *Kōgyō Jōrei*. Hence, in 1905, the mining-law *Kōgyō-Ho*, now in force, was promulgated, but it was only a rearrangement, without essential alterations, of the *Kōgyō Jōrei*. As to technical education, though the Tōkyō University supplied a considerable number of graduates, the demand of the mines was far from being satisfied, and the want was most keenly felt. So the government, as well as the people, have recently endeavored to establish educational institutions. The result has been the opening of the Kyōto Imperial University and the three government technical high-schools in Osaka, Kumamoto, and Sendai. Another mining high-school is now being built in the city of Akita by the joint efforts of the mine-owners and the government, besides one erected near Wakamatsu in Kyūshū by the private enterprise of K. Yasukawa. Lately the courses of lectures on mining and metallurgy have been opened in the Waseda University, founded by Count Okuma, and the Kyūshū Imperial University at the city of Fukuoka. The Kōshu-Gakkō or Foreman's School in Tōkyō, the Technical School of Fukuoka Prefecture, and others are worthy of note. The first, being the oldest school of this class, has the largest number of graduates. Also, mines such as the

Miike colliery and others, where primary mining-education is given, have recently increased. Thus, in the future, a sufficient number of engineers will be supplied, and it is expected that those who are advanced in technical knowledge will be obtainable at pleasure, according to the demand.

In 1908, the motive-power utilized in all the Japanese mines and collieries amounted to 275,477 h-p. including both steam- and water-power; but it is a conspicuous fact that there was not a single boiler at the time of the Restoration (1867) and in 1890 the steam- and water-power did not exceed 5,300 h-p. In future, it is expected that electricity generated by water will be utilized extensively, so that the amount of the motive-power will show correspondingly a great increase.

From the short description given above, it will be seen that the mining industry before the Restoration was insignificant—the mineral products in 1874 were valued at £154,690. The output had increased in 1908 to £11,638,667. The development of the industry is shown in Tables I. to IV., and the production of principal mines in 1908 in Tables V. to VIII. The principal causes of this great prosperity are found in the introduction of the modern civilization of Europe and America, at the time of Restoration of the present *régime*, with wise judgment and prudent solution, and in the revival, under the power of administration and education, of the industrial genius of the early Japanese, which has thus been directed in fruitful activity.

In reporting the results of the judgment of the Fifth National Exhibition at Osaka in 1903, Prof. W. Watanabe, member of the jury, said:

"After all, the reason why the mining industry has made an incomparably rapid progress depends before any thing else upon the wise administration of the government, to which may be added the following reasons:

"(1) The establishment of typical mining-works by the government, and the introduction of foreign technics; (2) the introduction of education regarding the mining industry with satisfactory results; (3) the handing over of the government mines to private operation; (4) the publication of mining-laws to meet the development of the mining industry; (5) the application of electricity generated by water-power as motive-power in the mining industry."

TABLE I.—*Production of Gold, Silver, Copper, and Lead in Japan*, 1874 *to* 1908.

Year.	Gold.		Silver.		Copper.		Lead.	
	Troy Oz.	Value.	Troy Oz.	Value.	Long Tons.	Value.	Long Tons.	Value.
1874	3,129	£6,415.2	87,890	£10,820.4	2,078.5	£76,882.0	.....	.....
1875	5,598	11,436.2	224,842	27,609.2	2,363.1	87,179.2	.....	.....
1876	7,147	14,569.0	280,892	34,634.9	3,135.3	115,907.4	.....	.....
1877	11,264	23,376.8	355,126	44,860.7	3,884.2	143,237.2	.....	.....
1878	8,764	18,143.2	318,017	40,300.4	4,194.1	155,630.6	.....	.....
1879	8,402	25,991.7	292,172	52,631.4	4,562.4	218,934.9	.....	.....
1880	9,925	37,486.1	332,406	78,277.7	4,600.8	270,196.7	.....	.....
1881	9,792	35,751.9	574,270	119,562.1	4,701.4	259,329.3	.....	.....
1882	8,736	23,618.7	558,783	95,496.1	5,533.4	258,981.3	.....	.....
1883	9,669	22,405.9	775,840	74,563.8	6,674.8	231,508.8	.....	.....
1884	8,830	21,424.6	736,321	86,974.6	8,758.3	245,296.6	.....	.....
1885	8,811	21,997.1	766,360	97,141.9	10,386.4	229,149.2	.....	.....
1886	14,937	31,441.1	1,083,057	117,814.7	9,630.9	233,909.0	.....	.....
1887	16,739	36,740.1	1,024,608	144,994.9	10,901.1	241,949.9	.....	.....
1888	20,230	43,728.2	1,374,113	147,655.2	13,179.3	475,862.8	.....	.....
1889	24,709	47,163.7	1,381,497	177,858.5	16,015.5	426,169.6	.....	.....
1890	23,362	49,410.6	1,699,029	209,341.6	17,849.5	509,708.6	.....	.....
1891	23,217	52,954.0	1,886,324	229,985.5	18,754.6	504,376.6	791	£7,860.5
1892	22,523	55,905.1	1,936,758	243,682.2	20,423.5	524,040.7	892	7,426.1
1893	23,676	61,130.6	2,226,825	273,714.8	17,750.8	492,413.3	1,089	10,273.0
1894	25,260	80,116.2	2,328,131	285,453.6	19,622.5	568,148.2	1,395	15,589.5
1895	40,898	106,213.3	2,323,673	254,470.8	18,834.4	720,484.1	1,904	21,247.7
1896	30,928	97,103.2	2,068,864	286,574.1	19,784.8	634,025.4	1,913	20,514.4
1897	33,617	120,758.4	1,745,657	191,116.0	20,091.5	783,494.0	754	8,391.4
1898	39,303	142,939.1	1,943,362	215,984.4	20,715.9	886,711.9	1,667	21,261.9
1899	58,654	238,003.8	1,805,879	215,648.1	23,920.6	1,456,549.5	1,946	28,495.8
1900	80,596	325,124.9	1,890,716	233,655.8	24,938.9	1,628,238.8	1,839	32,552.8
1901	101,683	446,846.1	1,760,158	211,681.9	26,990.0	1,625,244.2	1,765	24,640.9
1902	143,993	565,394.4	1,852,067	193,675.2	29,144.0	1,374,294.1	1,610	18,911.1
1903	139,023	554,027.7	1,884,162	196,603.3	32,111.0	1,720,213.9	1,689	20,416.8
1904	132,814	545,686.8	1,977,756	228,724.2	31,653.0	1,797,925.5	1,765	23,623.4
1905	148,645	612,903.3	2,678,511	330,983.9	34,975.0	2,366,333.1	2,224	32,189.6
1906	132,986	549,820.6	2,543,774	345,621.5	37,950.0	3,007,992.6	2,754	49,690.9
1907	134,153	554,740.7	3,091,022	424,601.8	39,556.0	3,372,887.7	3,015	56,863.6
1908	168,883	698,752.4	3,993,061	451,168.5	41,113.0	2,301,184.9	2,849	40,512.3

TABLE II.—*Production of Tin, Iron, Pyrite, and Antimony in Japan*, 1874 *to* 1908.

Year.	Tin.		Iron.		Iron Pyrite.		Antimony.	
	Long Tons.	Value.	Long Tons.	Value.	Long Tons.	Value.	Long Tons.	Value.
1874	.....	.....	4,817	£8,019.8	.....	.....	.....	.....
1875	.....	.....	3,383	5,646.8	.....	.....	.....	.....
1876	.....	.....	6,346	10,591.5	.....	.....	34	£193.9
1877	.....	.....	8,087	13,497.4	.....	.....	19	106.4
1878	.....	.....	10,006	16,701.7	.....	.....	16	974.9
1879	.....	.....	12,824	33,404.4	.....	.....	639	3,668.9
1880	.....	.....	15,889	47,885.2	.....	.....	494	2,836.5
1881	.....	.....	15,869	53,846.1	.....	.....	382	2,194.6
1882	.....	.....	12,070	38,261.3	.....	.....	2,432	16,312.5
1883	.....	.....	14,600	33,881.9	.....	.....	2,338	13,811.9
1884	.....	.....	11,672	27,134.2	.....	.....	1,446	12,558.1
1885	.....	.....	6,671	24,843.2	.....	.....	2,605	25,152.8
1886	.....	.....	13,546	21,766.8	.....	.....	2,342	19,621.7
1887	.....	.....	15,027	31,068.4	.....	.....	1,516	14,448.1
1888	.....	.....	18,003	43,164.1	.....	.....	1,232	17,509.9
1889	.....	.....	20,830	51,966.2	.....	.....	1,730	32,111.5
1890	.....	.....	22,059	50,453.9	.....	.....	2,055	43,491.6
1891	43	£229.2	17,038	28,879.5	240	£61.2	1,629	31,034.1
1892	40	2,395.6	19,442	41,774.7	2,220	1,365.0	1,317	18,358.7
1893	37	2,312.3	17,190	40,462.0	410	324.8	1,566	20,878.5
1894	38	2,494.8	19,225	57,169.6	5,045	1,088.9	1,538	24,885.7
1895	47	3,093.6	25,339	90,678.3	6,192	1,072.0	1,648	28,541.2
1896	46	2,650.9	26,762	96,215.1	8,700	908.9	1,315	23,028.4
1897	46	2,652.8	27,548	100,205.4	7,466	3,221.4	1,146	20,403.1
1898	42	2,271.4	23,238	83,116.7	8,543	2,783.1	1,211	21,331.4
1899	18	1,712.4	22,332	91,664.7	8,200	1,116.8	921	22,014.7
1900	12	1,181.1	24,468	95,528.6	15,826	2,586.6	420	12,184.8
1901	14	1,374.9	68,959	294,886.9	17,219	2,778.2	359	13,481.4
1902	18	1,861.0	63,462	296,168.3	18,190	2,824.2	603	13,451.9
1903	19	1,983.3	74,918	349,771.8	15,782	2,579.4	574	10,782.8
1904	24	2,607.3	74,310	338,328.0	24,363	5,308.9	417	8,374.4
1905	25	3,315.7	137,748	721,500.8	25,032	7,500.3	280	9,178.4
1906	23	4,089.1	181,608	846,495.7	35,212	9,399.5	296	22,862.6
1907	31	5,018.5	190,577	988,650.7	54,987	20,294.7	243	14,383.4
1908	25	3,507.6	183,110	917,003.5	33,156	17,145.9	194	5,347.8

TABLE III.—*Production of Zinc-Ore, Manganese-Ore, Phosphate-Rock, and Sulphur in Japan,* 1874 *to* 1908.

Year.	Zinc-Ore.		Manganese-Ore.		Phosphate-Rock.		Sulphur.	
	Long Tons.	Value.	Long Tons.	Value.	Long Tons.	Value.	Long Tons.	Value.
1874	.....	.....	.....	.....	.....	.....	569	£1,042.4
1875	.....	.....	.....	.....	.....	.....	574	1,051.4
1876	.....	.....	.....	.....	.....	.....	1,368	2,508.0
1877	.....	.....	.....	.....	.....	.....	1,297	2,377.6
1878	.....	.....	.....	.....	.....	.....	2,107	3,861.8
1879	.....	.....	.....	.....	.....	.....	1,714	3,115.9
1880	.....	.....	.....	.....	.....	.....	1,166	2,139.9
1881	.....	.....	.....	.....	.....	.....	684	1,253.2
1882	.....	.....	.....	.....	.....	.....	3,331	4,481.2
1883	.....	.....	.....	.....	.....	.....	6,794	14,042.1
1884	.....	.....	.....	.....	.....	.....	4,189	15,990.9
1885	.....	.....	.....	.....	.....	.....	4,846	8,374.0
1886	.....	.....	.....	.....	.....	.....	6,312	9,852.9
1887	.....	.....	.....	.....	.....	.....	10,555	14,433.3
1888	.....	.....	.....	.....	.....	.....	19,199	21,816.6
1889	.....	.....	.....	.....	.....	.....	16,527	29,369.1
1890	.....	.....	.....	.....	.....	.....	21,184	30,158.4
1891	.....	.....	3,156	£6,053.2	.....	.....	21,469	34,000.4
1892	.....	.....	4,913	1,873.8	.....	.....	20,055	41,245.2
1893	.....	.....	3,958	11,593.6	.....	.....	23,387	44,079.8
1894	.....	.....	13,064	8,391.0	.....	.....	18,360	30,206.1
1895	.....	.....	16,753	7,820.1	.....	.....	15,204	24,208.2
1896	.....	.....	17,559	12,526.9	.....	.....	12,255	25,113.7
1897	.....	.....	15,097	8,764.2	.....	.....	13,297	33,593.1
1898	.....	.....	11,256	8,029.1	.....	.....	10,644	29,497.8
1899	.....	.....	11,098	7,935.2	.....	.....	10,585	28,865.0
1900	.....	.....	15,498	16,303.0	.....	.....	14,858	33,515.6
1901	.....	.....	15,928	10,846.4	.....	.....	17,806	43,091.8
1902	.....	.....	10,638	6,701.0	193	£120.1	19,502	48,285.8
1903	.....	.....	5,489	3,737.8	188	120.2	24,402	61,830.3
1904	.....	.....	4,236	3,603.9	13	16.2	28,141	65,116.8
1905	.....	.....	13,723	8,176.6	1,494	1,134.2	25,286	60,306.6
1906	13,998	£20,824.1	12,572	8,282.7	3,090	1,842.1	28,521	62,974.1
1907	19,358	32,154.8	20,153	14,410.1	1,693	740.4	33,917	81,728.4
1908	18,010	30,877.5	10,897	8,259.4	728	730.2	34,645	79,269.1

TABLE IV.—*Production of Graphite, Coal, Petroleum, etc., in Japan,* 1874 *to* 1908.

Year.	Graphite.		Coal.		Petroleum.		Other Products.	Total Mineral Production.
	Long Tons.	Value.	Long Tons.	Value.	U. S. Barrels.	Value.	Value.	Value.
1874	.....	.....	204,864	£49,065.1	3,499	£2,444.6	.....	£154,689.5
1875	.....	.....	558,288	114,838.8	5,489	3,835.2	.....	251,596.8
1876	.....	.....	537,011	111,998.5	9,267	6,276.8	.....	296,680.0
1877	.....	.....	491,835	103,649.0	12,628	8,030.5	.....	339,135.6
1878	.....	.....	669,806	146,797.5	21,500	15,022.7	.....	397,432.8
1879	.....	.....	845,057	161,395.3	28,199	19,703.5	.....	518,846.0
1880	.....	.....	865,201	212,270.5	30,651	21,417.2	.....	672,509.8
1881	.....	.....	911,720	225,233.3	20,138	14,070.5	.....	711,241.0
1882	.....	.....	915,676	265,982.2	23,311	17,642.5	£12,550.7	733,326.5
1883	.....	.....	987,818	236,385.1	24,613	10,666.8	10,597.3	647,863.6
1884	.....	.....	1,123,330	242,170.3	7,063	10,796.4	7,307.2	669,652.7
1885	.....	.....	1,274,775	266,238.5	8,324	9,849.6	7,172.7	689,819.0
1886	.....	.....	1,354,190	280,360.3	15,326	13,691.1	11,515.4	739,943.0
1887	.....	.....	1,720,909	307,701.0	10,034	12,629.8	16,027.6	819,993.1
1888	.....	.....	1,893,970	409,060.2	45,006	13,860.2	17,928.6	1,190,585.8
1889	.....	.....	2,353,849	541,723.4	63,490	25,097.7	15,035.8	1,346,495.5
1890	.....	.....	2,589,997	623,366.2	61,817	22,147.8	15,151.0	1,553,309.7
1891	241	£915.5	3,129,581	657,399.7	63,618	20,702.9	1,452.2	1,575,904.5
1892	59	348.6	3,129,409	606,870.5	82,833	22,530.2	35.3	1,567,851.7
1893	27	241.6	3,271,244	625,744.8	106,983	20,649.3	1,155.8	1,604,974.2
1894	1,066	2,515.7	4,214,253	1,020,511.1	172,711	26,163.5	1,701.5	2,124,435.4
1895	428	781.0	4,718,914	1,275,787.5	169,873	38,227.8	1,701.5	2,574,327.1
1896	210	2,594.8	4,946,568	1,187,423.3	236,819	35,160.7	2,004.8	2,425,844.6
1897	382	6,168.9	5,131,628	1,909,692.1	262,751	34,821.9	596.5	3,223,879.2
1898	339	6,238.6	6,640,468	2,709,954.7	319,015	34,896.2	314.6	4,165,330.9
1899	52	969.6	6,653,476	2,306,362.5	539,098	104,369.4	1,737.9	4,505,481.4
1900	92	2,385.5	7,362,891	2,474,326.5	871,740	194,201.3	2,139.2	5,053,924.5
1901	86	1,743.3	8,879,511	3,084,254.6	1,117,995	227,882.1	1,274.5	5,990,027.2
1902	95	1,984.2	9,656,295	3,255,718.7	997,543	207,784.0	475.8	5,987,649.8
1903	111	2,199.4	10,021,893	2,925,307.2	1,210,340	281,936.2	316.6	6,132,127.2
1904	212	3,708.8	10,649,026	2,947,678.7	1,220,744	278,060.6	271.8	6,249,036.2
1905	204	2,651.8	11,467,845	4,050,127.9	1,352,574	286,948.1	371.5	8,504,621.9
1906	138	2,444.9	12,892,721	6,348,445.7	1,571,367	317,901.1	867.8	11,599,565.0
1907	101	1,046.9	13,736,182	6,044,584.6	1,727,298	527,749.6	1,861.2	12,141,716.6
1908	147	1,726.3	14,761,476	6,416,698.4	1,872,592	658,598.4	7,885.0	11,638,667.2

TABLE V.—*Production of Principal Mines in* 1908.

GOLD, SILVER, COPPER, AND LEAD.

Mines.	Prefectures.	Concessionaires.	Output. Gold.	Silver.	Copper. Metal.	Copper. Ore.	Lead.	No. of Laborers.
			Troy Ounces.	Troy Ounces.	Long Tons.	Long Tons.	Long Tons.	
Poropets	Hokkaidō	Oda Ryoji	6,559.2	6,629.5	69.0			267
Shiribeshi	Hokkaidō	Tokunaga Shigeyasu	1,033.2	14,787.1				149
Kosaka	Akita	Fujita & Co.	10,490.8	1,110,707.2	7,086.2		372.2	7,642
Tsubaki	Akita	Takeda Kyōsaku		1,244,572.8	273.9			1,365
Ani	Akita	Furukawa Mining Co.		44,948.3	1,284.6			3,560
Osaruzawa	Akita	Mitsubishi & Co.	412.7	25,219.0	1,358.4			1,502
Komaki	Akita	Mitsubishi & Co.						119
Arakawa	Akita	Mitsubishi & Co.			732.2			1,747
Hisan-ichi	Akita	Mitsubishi & Co.		34,508.0	689.7			753
Furokura	Akita	Furukawa Mining Co.			578.5			734
Innai	Akita	Furukawa Mining Co.	967.9	94,757.9	152.7			347
Tokitō	Akita	Fujita & Co.			279.0	4,078.3		375
Hanaoka	Akita	Ishida Kumakichi				4,203.2		90
Daiji	Akita	Fujita & Co.			74.5			117
Matsuoka	Akita	Fujita & Co.	365.9	11,150.8				270
Okuzu	Akita	Mitsubishi & Co.	48.0	89.0		1,914.7		121
Midzusawa	Iwate	Furukawa Mining Co.			255.6			733
Kamaishi	Iwate	Tanaka Chōbei	2,013.4		47.5	3,288.1		4,555
Washinosu	Iwate	Tameda Buntarō	1,346.2	83.6				253
Oarazawa	Iwate	Saitō Tatsugorō			73.8			78
Unekura	Iwate	Satō Jirō			74.2			126
Shishiori	Miyagi	Tokunaga Shigeyasu	2,041.7	569.2				150
Nagamatsu	Yamagata	Furukawa Mining Co.		12,993.2	259.2			504
Ōtori	Yamagata	Furukawa Mining Co.		8,988.4	122.9			378
Karatoya	Yamagata	Karatoya Mining Co.			1.4	429.8		101
Yoshino	Yamagata	Yokota Ichisaku and others				143.2		65
Kanō	Fukushima	Kanō Mining Co.	781.1	91,624.7	974.0			2,181
Handa	Fukushima	Godai Riusaku	656.9	21,249.4				112
Yakuki	Fukushima	Yakuki Mine Co.			113.5			109
Sado	Niigata	Mitsubishi & Co.	13,740.0	113,651.9	3.7			1,238
Kusakura	Niigata	Furukawa Mining Co.			237.7			489
Hirotani	Niigata	Furukawa Mining Co.			61.8			139
Ashio	Tochigi	Furukawa Mining Co.		75,082.4	6,972.4			7,274
Kobyaku	Tochigi	Kobyaku Mine Co.			154.0			150
Hitachi	Ibaraki	Kuhara Fusanosuke	1,558.4	30,615.5	1,871.5			1,013
Takara	Yamanashi	Mitsubishi & Co.			109.7			95
Kune	Shizuoka	Furukawa Mining Co.				43,018.2		438
Ōmatsuyama	Shizuoka	Kurotaki Chōjiro	512.4	3,608.9				44
Kamioka	Gifu	Mitsui Mining Co.	246.4	169,529.8	38.9		2,129.8	2,251
Hiragane	Gifu	Yokoyama Takaoki		38,996.3	509.7			824
Takane	Gifu	Asada Sanemon		14,168.3	141.6			227
Hatasa	Gifu	Okunō Mining Co.		25,327.9	44.7			182
Ogoya	Ishikawa	Yokoyama Takatoshi			685.4			1,143
Yūsenji	Ishikawa	Yoshinotani Coal Mine Co.			610.5			880
Kuratani	Ishikawa	Kuratani Mine Co.	2,415.7	38,535.1	2.5		184.0	352
Kanahira	Ishikawa	Murata Sukematsu	1,066.4					424
Ate	Ishikawa	Moriyama Saichi			60.4			160
Togi	Ishikawa	Kinoshita Ryō and others	697.0	624.5				138
Omodani	Fukui	Mitsubishi & Co.	7.2	28,689.1	182.8			360
Takamasa	Shiga	Osaka Mining Co.		848.3	9.9			23
Iimori	Wakayama	Nakae Tanezo			75.3			70
Kyōsei	Nara	Tanaka Ginnosuke				4,875.6		130
Ikuno	Hyōgo	Mitsubishi & Co.	3,881.0	211,865.5	1,152,1	4,215.1		1,949
Kanasaka	Hyōgo	Hayashi Heizō and others			37.9			79
Karatani-Kawakami	Hyōgo	Tsuchiya Minekichi		3,072.0	27.9			78
Tada	Hyōgo	Hori Tōjurō		4,656.2	18.7		0.5	
Ōmori	Shimane	Fujita & Co.	1,689.6	96,153.6	314.2		4.1	675
Hōmanzan	Shimane	Hori Tōjurō			270.9			491
Sasagatani	Shimane	Hori Tōjurō		12,210.0	144.3			296
Dōgamaru	Shimane	Hori Tōjurō		9,504.0	56.7			166
Kuki	Shimane	Hori Tōjurō		20,291.6			102.5	105
Wanibuchi	Shimane	Wanibuchi Mining Co.			11.1	1,254.2		111
Yoshioka	Okayama	Mitsubishi & Co.	138.8	60,590.4	778.5			1,273
Obiye	Okayama	Sakamoto & Co.			736.4			813

TABLE V.—*Production of Principal Mines in* 1908.—*Continued.*

GOLD, SILVER, COPPER, AND LEAD.—Continued.

Mines.	Prefectures.	Concessionaires.	Output.					No. of Laborers.
			Gold.	Silver.	Copper.		Lead.	
					Metal.	Ore.		
			Troy Ounces.	Troy Ounces.	Long Tons.	Long Tons.	Long Tons.	
Mihara	Okayama	Utsunomiya Mining Co.				1,616.8		177
Kokusei	Okayama	Suzuki Ekiji			116.5	273.7		68
Konjō	Okayama	Sakata Mitsugi		8,559.9	41.9			182
Yamate	Okayama	Nomura Chōbei		2,380.3	31.1			66
Hisaki	Okayama	Kusakabe Toraji				287.2		50
Naganobori	Yamaguchi	Hori Tōjurō		6,693.1	115.2			176
Yakuōji	Yamaguchi	Kawabe Kurasaburo			53.2	871.1		191
Ōta	Yamaguchi	Ikuta Kunizo and others	5.3	2,389.4	64.1			133
Kitsunezuka	Yamaguchi	Kuwabara Masa and others			24.0			41
Mochibe	Tokushima	Shima Tokuzō				13,816.9		335
Higashiyama	Tokushima	Shima Tokuzō				2,963.3		2,687
Besshi	Ehime	Sumitomo Kichizaemon			5,173.3			3,477
Chihara	Ehime	Yabuuchi Senzō			170.5			165
Kanayama	Ehime	Mitsubishi & Co.				7,841.8		224
Nishinokawa	Ehime	Sumitomo Kichizaemon			87.8			161
Kuchō	Ehime	Seike Kumeichirō				1,988.2		195
Hirabaye	Ehime	Yamashita Kichiro				4,136.0		75
Ōmine	Ehime	Shiraishi Watarō				1,956.0		123
Kajitani	Ehime	Shiraishi Watarō				2,097.1		73
Takaura	Ehime	Fujino Kamenosuke				1,615.1		35
Ōye	Ehime	Yano Sōzaburō				603.6		69
Nagamine	Ehime	Ueda Seiichi and others				305.1		26
Yano	Fukuoka	Yano Tomokichi	781.9	557.0				218
Taiōno	Ōita	Nangō Tokunosuke and others	3,099.4	2,035.2				226
Mizobe	Ōita	Kojima Tetsutarō	352.0	231.7				95
Iwaya	Kumamoto	Noda Kichibei			71.5			57
Hibira	Miyazaki	Naito Seikyo			870.3			1,021
Makimine	Miyazaki	Mitsubishi & Co.			541.6			866
Yamagano	Kagoshima	Shimazu Tadashige	12,167.5	26,735.3				2,219
Ushio	Kagoshima	Ushio Gold Mine Co.	8,932.6	3,426.6				477
Okuchi	Kagoshima	Iwatsuki Naohiko and others	4,766.8	2,115.2				424
Serigano	Kagoshima	Shimazu Tadashige	2,496.4	9,714.6				618
Nitabira	Kagoshima	Nagai Eikichi and others	1,325.5	2,847.7				198
Urushi	Kagoshima	Kamimura Ryōsuke and others	1,181.6	4,254.4				243
Fuke	Kagoshima	Asano Sōichirō	884.8	701.3				87
Bezaiten	Kagoshima	Hitaka Shōkō and others	179.3	24,271.6				98
Otani	Kagoshima	Satsunan Mining Co.	663.5	363.0				212
Kago	Kagoshima	Horinouchi Shōuemon	300.6	202.1				310
Hashima	Kagoshima	Horinouchi Shōuemon	145.7	693.1				17
Kinkaseki	Taiwan (Formosa)	Tauaka Chōbei	33,353.3	25,558.0	319.0			904
Botankō	Taiwan (Formosa)	Kimura Kintarō	9,193.7	3,873.1				479
Zuihō	Taiwan (Formosa)	Fujita & Co.	8,961.9	3,984.4				550

TABLE VI.—*Production of Principal Mines in* 1908.

ANTIMONY, IRON, MANGANESE-ORE AND SULPHUR.

Mines.	Prefectures.	Concessionaires.	Output. Antimony.	Output. Iron.	Output. Manganese-Ore	Output. Sulphur.	No. of Laborers.
			Long Tons	Long Tons.	Long Tons.	Long Tons.	
Oshino	Hokkaido	Oshino Kyō			7,455.9	9,457.8	469
Kobui	Hokkaido	Senshō Kōgyō Co				5,574.9	259
Iwaonupuri	Hokkaido	Mitsui Mining Co			1,455.5	3,002.0	286
Kumadomari	Hokkaido	Endō Kichihei				2,006.9	147
Shikabe	Hokkaido	Oshima Mining Co				1,790.7	143
Uguisuzawa	Iwate	Satō Seibei and others				592.1	69
Kamaishi	Iwate	Tanaka Chōbei		36,662.4			4,555
Sennin	Iwate	Sennin Iron Foundry Co		1,018.6 (ore) 2,885.7			958
Numajiri	Fukushima	Japanese Sulphur Co				5,284.0	218
Ichinokawa	Ehime	Ichinokawa Mining Co.	158.3				259
Kujusan	Ōita	Hiromi Nisaburō				1,328.1	56
Kano	Yamaguchi	Sunohara Kumajirō and others	(ore) 259.2				65
Iwōjima	Kagoshima	Hiromi Nisaburō				607.5	76

TABLE VII.—*Production of Principal Petroleum-Fields in* 1908.

Oil-Fields.	Prefectures.	Concessionaires.	Output* Barrels (42 Gallons)	No. of Laborers.
Koguchi	Niigata	Hōden Oil Co	247,151.6	143
Niitsu	Niigata	Nippon Oil Co	170,230.7	254
Kanatsu	Niigata	Hōden Oil Co	129,325.0	77
Kanatsu	Niigata	Nakano Kanichi & others	86,018.2	95
Asahi	Niigata	Uchida Sansei	54,340.0	10
Asahi (19)	Niigata	Chūō Oil Co	41,001.1	81
Kumazawa	Niigata	Hōden Oil Co	31,421.6	82
Asahi (349)	Niigata	Chūō Oil Co	21,349.0	86
Koguchi	Niigata	Fuji Oil Co	19,629.5	13
Asahi 5th	Niigata	Nakano Chūtaro	17,008.0	43
Higashiyama	Niigata	Hōden Oil Co	234,477.3	1,522
Urase	Niigata	Nippon Oil Co	49,886.4	54
Nagamine	Niigata	Nippon Oil Co	331,920.5	706
Nagamine	Niigata	Hōden Oil Co	163,690.9	140
Kamada	Niigata	Hōden Oil Co	47,129.5	55
Maki	Niigata	Nippon Oil Co	31,812.5	239
Maki	Niigata	Hōden Oil Co	31,343.2	160

* The output is calculated as crude oil.

TABLE VIII.—*Production of Principal Coal-Mines in* 1908.

Mines.	Prefectures.	Concessionaires.	Output.	No. of Laborers.
			Long Tons.	
Yūbari 1st	Hokkaido	Hokkaido Colliery & S. S. Co	488,126.9	5,008
Soracpi	Hokkaido	Hokkaido Colliery & S. S. Co	247,379.5	1,957
Shin-Yūbari	Hokkaido	Ishikari Coal Co. & others	165,390.8	1,583
Horonal	Hokkaido	Hokkaido Colliery & S. S. Co	187,269.6	1,238
Yūbari 2nd	Hokkaido	Hokkaido Colliery & S. S. Co	116,726.2	913
Ikushumpets	Hokkaido	Hokkaido Colliery & S. S. Co	76,847.4	718
Pompets	Hokkaido	Nippon Kōgyō Co	83,015.3	395
Uchinogō	Fukushima	Iwaki Coal Mine Co	198,936.3	1,408
Onoda	Fukushima	Iwaki Coal Mine Co	190,562.4	1,194
Iriyama	Fukushima	Iriyama Coal-Mining Co	194,742.5	1,402
Yoshima	Fukushima	Yoshima Coal-Mining Co	126,826.9	1,348
Ōjō	Fukushima	Ōjō Coal-Mining Co	68,694.2	441
Sansei	Fukushima	Sansei Coal-Mining Co	69,698.1	548
Ibaraki Saitan	Ibaraki	Ibaraki Coal-Mining Co	69,733.6	593
Ibaraki Muentan	Ibaraki	Ibaraki Anthracite Mining Co	59,042.2	622
Ōmine	Yamaguchi	Navy Department	96,885.7	974
Okinoyama	Yamaguchi	Watanabe Yūsaku	53,282.7	508
Kata	Yamaguchi	Noda Kichibei & others	58,826.8	149
Ōtsuji 4th	Fukuoka	Kaijima Tasuke	197,115.8	1,694
Ōtsuji	Fukuoka	Kaijima Tasuke	167,950.6	1,447
Miyoshi	Fukuoka	Miyoshi Tokumatsu	120,653.1	731
Iwasaki	Fukuoka	Iwasaki Kumekichi	85,417.7	486
Nakatsuru	Fukuoka	Itō Denyemon	57,656.4	371
Arate	Fukuoka	Itō Denyemon	54,450.9	558
Onoura	Fukuoka	Kaijima Tasuke	771,290.8	6,195
Shinnyu	Fukuoka	Mitsuaishi & Co	426,621.0	4,945
Meiji	Fukuoka	Meidi Mining Co	419,793.6	3,186
Shiogashira and Shakanoo	Fukuoka	Furukawa Mining Co	373,198.0	2,705
Mitsui-hondo	Fukuoka	Mitsui Mining Co	195,914.8	3,084
Gotoku Kaigun	Fukuoka	Navy Department	213,383.4	1,532
Koyanose (508)	Fukuoka	Egi Iwakichi	83,141.3	553
Koyanose (101)	Fukuoka	Egi Iwakichi	46,730.8	180
Namazuda	Fukuoka	Mitsubishi & Co	271,836.0	2,207
Futase	Fukuoka	Government Steel Works	354,935.9	2,629
Yoshio	Fukuoka	Asō Takichi	243,306.0	1,792
Mitsui-yamano	Fukuoka	Mitsui Mining Co	177,693.8	2,320
Tadakuma	Fukuoka	Sumitomo Kichizaemon	168,959.3	1,697
Mamena	Fukuoka	Asō Takichi	150,178.0	1,076
Aida	Fukuoka	Nakano Tokujiro & others	97,310.8	452
Kamiyamada	Fukuoka	Mitsubishi & Co	93,850.4	1,021
Muta	Fukuoka	Itō Denyemon	91,832.8	758
Shimoyamada	Fukuoka	Furukawa Mining Co	105,335.9	1,103
Hirayama	Fukuoka	Tajima Shinobu	32,720.7	537
Honami	Fukuoka	Ishida Osamu	16,120.2	166
Mitsui-tagawa	Fukuoka	Mitsui Mining Co	536,290.0	4,460
Kanada	Fukuoka	Kanada Mining Co	290,978.6	2,442
Ōtō	Fukuoka	Buzen Coal-Mining Co	171,546.0	1,305
Akaike	Fukuoka	Meidi Mining Co	172,952.2	1,444
Kōkoku	Fukuoka	Meidi Mining Co	127,083.8	1,454
Hōjō	Fukuoka	Mitsubishi & Co	96,648.4	1,188
Miyazaki-hōshu	Fukuoka	Miyazaki Giichi	106,002.3	929
Mineji	Fukuoka	Kashiwagi Kanpachiro	66,569.3	1,450
Honsoeda	Fukuoka	Masuya Hikosuke	60,447.8	353
Miyao	Fukuoka	Kurachi Jiichi	66,788.8	362
Soeka	Fukuoka	Abe Yasujirō	53,054.4	645
Kigyo-Komatsu-Gotōji	Fukuoka	Kurachi Shigehiko	64,765.3	556
Kaharu	Fukuoka	Kuwahara Masa	47,257.3	669
Tsubakuro	Fukuoka	Kaijima Tasuke	81,214.2	835
Shimbaru Kaigun	Fukuoka	Navy Department	81,921.1	1,318
Sasakuri	Fukuoka	Yonezawa Tokichi	60,593.2	354
Ueki	Fukuoka	Okada Sango	53,389.9	334
Miike	Fukuoka	Mitsui Mining Co	1,513,388.6	9,976
Yoshinotani	Saga	Yoshinotani Coal-Mining Co	251,770.7	3,160
Ochi	Saga	Mitsubishi & Co	161,468.0	2,476
Kishidake	Saga	Koga Seijiro & others	83,017.3	1,188
Akasakaguchi	Saga	Takatori Koreyoshi	124,462.9	1,547
Yunokibaru	Saga	Kaijima Tasuke	87,350.7	1,149
Kishima, 2nd	Saga	Tajima Shinobu	61,926.9	1,171
Kishima	Saga	Hieda Ichirōji & others	71,243.3	1,892
Takashima	Nagasaki	Mitsubishi & Co	184,816.9	2,693
Matsushima	Nagasaki	Koga Shun-ichi	78,753.7	1,326
Matsura	Nagasaki	Matsura Coal-Mining Co	80,719.2	486

Notes on the Laramie Tunnel.

BY DAVID W. BRUNTON, DENVER, COLO.

(San Francisco Meeting, October, 1911.)

MINE-DRAINAGE and the ever-increasing demand for water on the plains have within the past few years necessitated the driving of a great number of adits and tunnels, including many of considerable size and great length. In most instances, existing conditions have called for extreme rapidity of execution; and this necessity, coupled with the high altitudes at which most of the work has been carried on, excessive freight-rates, and difficulty of access, has involved many complex and interesting problems.

One of the most rapidly executed undertakings of this class is the Laramie tunnel, recently completed for the Laramie-Poudre Reservoirs & Irrigation Co. This tunnel is driven through Green ridge, a spur of the Continental Divide, which separates the Laramie river from the Cache-la-Poudre in Laramie county, Colo. Green ridge is extremely persistent, and further northward in Wyoming is known as Sherman hill, the highest point on the Union Pacific railway. At the point where the tunnel is driven, two opposing bends bring these streams within 2.5 miles of each other, with a difference in elevation of 500 ft. The rock exposed on the banks of the two rivers does not differ materially in hardness; and, without a geological study of the conditions involved, it would appear that the difference in the depth of erosion of the two streams is due principally to the greater volume of water at this point in the Cache-la-Poudre, which has a drainage-area above the tunnel-portal of 110 sq. miles, as against 30 sq. miles of natural drainage for the Laramie. To increase the water-supply at the tunnel, however, this latter amount has been augmented by two side collection-canals, which bring the available drainage-area up to 84 square miles.

The nearest railroad shipping-point to the eastern portal is

Fort Collins, Colo., 40 miles to the east in an air-line; and some idea of the difficulties of access and transportation may be gathered from the fact that the wagon-road between the two points is 65 miles long, and the freight-rate $22 per ton. The western portal was reached from Laramie City, Wyo., over a very hilly wagon-road 74 miles long, on which the freight-rate varied from $20 to $25 per ton, according to the season and the condition of the road.

The tunnel was driven for the purpose of conveying through the divide 800 cu. ft. of water per second, which is to be diverted from the Laramie and discharged into the Cache-la-Poudre, where it will serve to augment the water-supply of the latter stream and enable it, for a time at least, to meet the demand for water in the immensely-rich irrigation-district which it serves.

Fig. 1 shows the profile of the tunnel, which was run on a 1.7-per cent. grade, with an elevation at the intake of 8,580 ft., and a length from portal to portal of 11,306 ft. The open-cut approach on the eastern end was 45 ft., and on the western approach 1,200 ft. long, making a total length of cutting on the floor-line of 12,551 feet.

Fig. 2 shows a cross-section of the tunnel, which, as shown, is 7 ft. 6 in. in height by 9 ft. 6 in. in width, although, to save time and expense in trimming, it was generally run 8 by 10 ft. The rock is very hard, tough granite; and this section was selected, instead of the ordinary form of tunnel, with a view to diminishing the cost and expediting the work of driving; since the expense of cutting out the corners to the usual rectangular form would have been very considerable. Moreover, should soft ground be encountered anywhere in the course of the tunnel, the arched form selected would add greatly to the strength of the opening.

Work on the power-plant for operating the tunnel was begun Dec. 1, 1909, and continued throughout the winter, although progress was greatly impeded by intense cold and heavy snowfall.

The hydro-electric power-plant was erected on the west bank of the Cache-la-Poudre, nearly opposite the eastern portal, and the intake of the pipe-line was located at an exceedingly good natural site nearly 2 miles farther up the stream. The dam

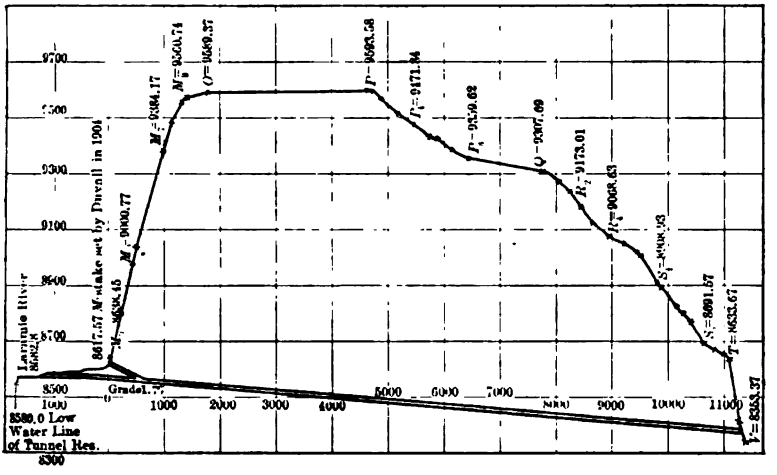


FIG. 1.—PROFILE OF THE LARAMIE RIVER TUNNEL.

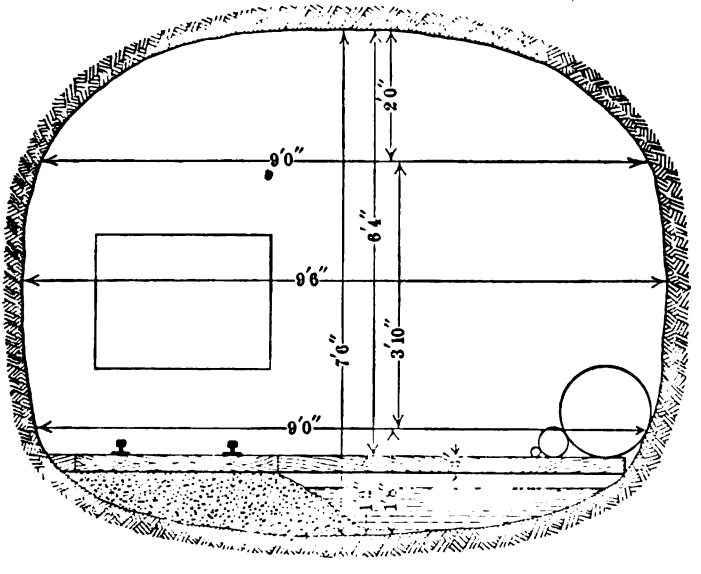


FIG. 2.—CROSS-SECTION OF THE LARAMIE RIVER TUNNEL, SHOWING POSITION OF TRACKS, DRAIN, VENTILATING, COMPRESSED-AIR AND WATER-PIPES.

was of the usual log-type, 10 ft. high, faced with two thicknesses of 2-in. plank, and fitted with suitable intake-screens, penstock, and gate. The water was conveyed from the dam to the power-plant through 8,500 ft. of 22-in. wooden pipe, bound together with round steel bands so as to stand the pressure-head at all points, with a factor of safety of 3.5. The bands were all dipped or painted with two coats of paint. The total head at the water-wheel was 268 ft. and the effective head 200 ft. Relief-valves were provided along the entire length of the pipe to prevent collapse in the case of sudden emptying. The foundations for the machinery, and the waste-ways from the water-wheels, were all of concrete. The power-plant consisted of:

One 48-in. Pelton double-nozzle wheel, developing, at 245 rev., 250 b.h-p., used for driving the generator. The governor was a special Pelton oil-governor of the piston-type, controlling Pelton stream-deflectors attached to the water-wheel nozzles.

One 48-in. Pelton single-nozzle water-wheel; capacity, 130 h-p. at 245 rev. The speed-control of this wheel, known as the Pelton special air-control mechanism, consists of a compressed-air cylinder with suitable connections to the nozzle-deflector. The flow of air to the cylinder is controlled by a pilot-valve with a safety counter-balance. This special air-control stops the operation of the water-wheel when the pressure in the air-receiver reaches the normal, and whenever the pressure drops 8 or 10 lb. it automatically starts the water-wheel; the function of the counter-balance being to stop the wheel in case of a rupture in the air-pipe or the tank. This device, while exceedingly simple, gave the utmost satisfaction and shut down the water-wheel whenever the demand for air stopped, thus greatly reducing the wear and tear on the compressor.

Electricity for power and light was supplied by a General Electric type A. T. B., 150-kw., 600-rev. per. min., 2,300-volt, A. C., 3-phase, 60-cycle generator.

Compressed air for the east portal was furnished by a belt-driven Ingersoll-Rand Imperial compressor, type N., 17- and 10-in. cylinders, 14-in. stroke, placed in the power-house; and air for the west portal was supplied by a duplicate compressor, electric-driven.

Ventilation in the heading from the eastern portal was fur-

nished by a Connersville special heavy-duty type of exhauster, belt-driven, having a displacement of 13 cu. ft. per rev. and a normal speed of 250 rev. per minute.

Ventilation in the heading from the western portal was provided by a duplicate exhauster, electric-driven; current for this purpose and for operating the air-compressor and electric hoist being carried over the mountain from the generating-station on the Cache-la-Poudre by a 3,300-volt power-line. Both exhausters were arranged with by-passes so that they could be used either for exhausting from the face or blowing air into it, as desired.

To operate the incline at the western portal, an electric hoist was installed which had a capacity of 5,000 lb., raised 120 ft. per min., the hoist being gear-connected to a General Electric 440-volt, 60-cycle, 3-phase, 25-h-p. motor, operated by a street-railway type controller and resistance.

The western, or intake, portal of the tunnel was necessarily near the level of the Laramie river channel; and as the bank immediately above it showed, by springs and seepage, the presence of a very considerable amount of water, it was deemed best to avoid tapping this water-bearing area by beginning work above it and running down to the grade of the tunnel with a 25-per cent. incline 188 ft. long through rock which promised to be nearly, if not quite, free from water.

Pending the completion of the power-plant, work was commenced with pneumatic drills, supplied with air from small steam-operated Norwalk compressors, at the eastern portal on Dec. 25, 1909, and on the incline at the west end of the tunnel on Jan. 16, 1910. At first Leyner No. 7 water-drills were used in both headings; but, although the rate of drilling was satisfactory, the repairs were high, and the desirability of a heavier and more sturdy drill became so apparent that Leyner designed and supplied to the contractor, in July, 1910, a very much improved and heavier drill, now catalogued and sold as Model No. 8.

In order to obtain, on the completion of the tunnel, as much salvage as possible from the sale of machinery and tools, everything purchased was of the very best quality, and, wherever possible, standard patterns and sizes were ordered. For this reason the tunnel-cars (shown in Figs. 3, 4, 5, and 6) were of

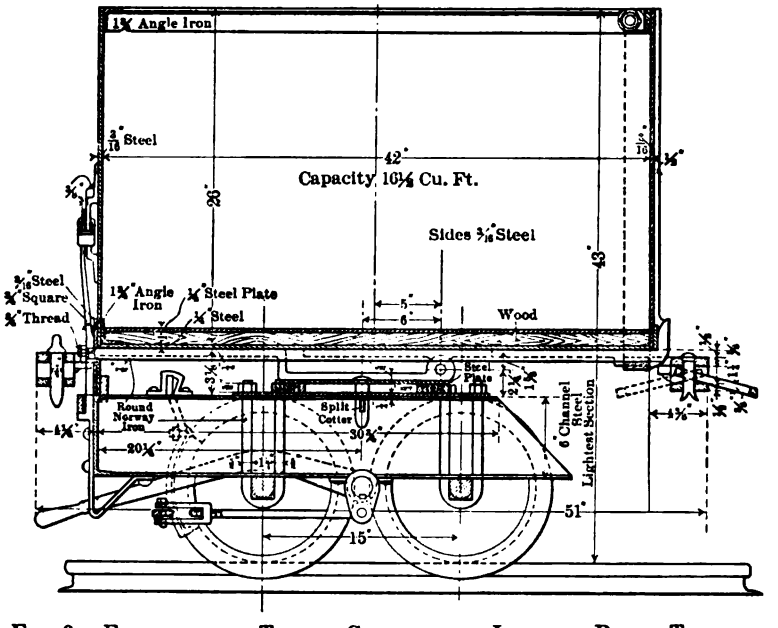


FIG. 3.—ELEVATION OF TUNNEL-CAR USED IN LARAMIE RIVER TUNNEL.

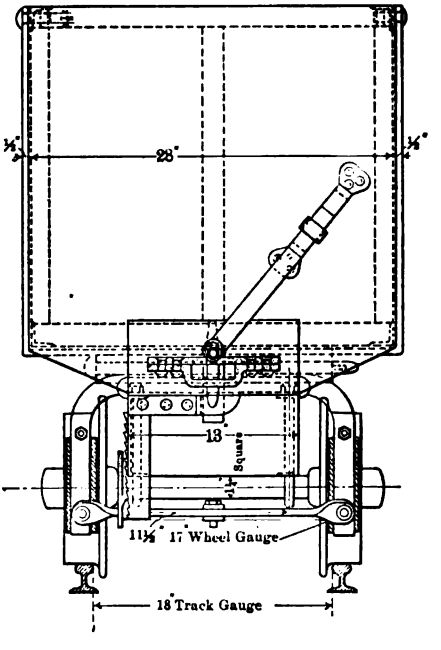


FIG. 4.—END-VIEW OF TUNNEL-CAR.

the standard mining type, fitted with brakes and equipped with turn-tables permitting either side- or end-dumping. The wheels were of the well-known Cowenhoven Tunnel pattern, with closed hub ends, as illustrated in Fig. 6, which also shows very clearly the brake-mechanism.

These wheels are lubricated by filling the hub recess with non-fluid oil, forced in from a "gun" through an opening closed by a 0.25-in. pipe-plug.

As the system of car-handling in the tunnel-headings necessitated throwing all of the cars over on their sides once, and nine-tenths of them twice, on each trip, the connections between the trucks and bodies of the cars were carefully planned

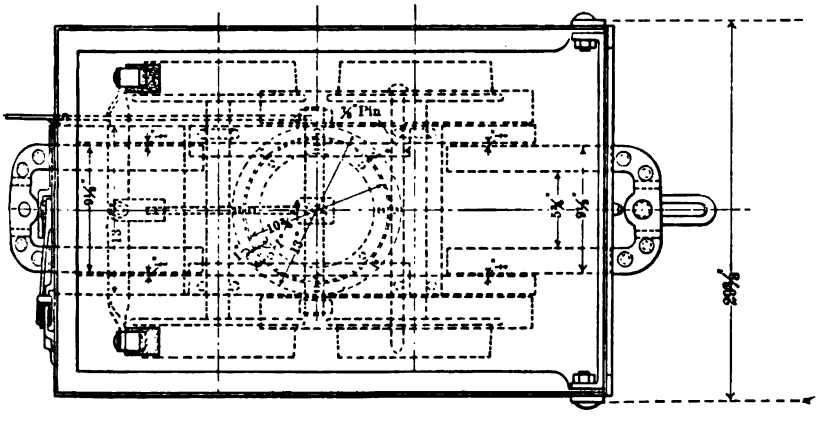


FIG. 5.—PLAN OF TUNNEL-CAR.

and made unusually strong. The turn-tables were fitted with two concentric rings, Figs. 3 and 5, and the locking-mechanism for securing the bodies to the trucks was so designed that when the releasing-lever was fastened in place the cars were as rigid as if the bodies were riveted to the axles.

Owing to the great length and number of drills used, they were hauled into the tunnel on special cars, Fig. 7, 10 ft. long, and provided with compartments to keep separate the different lengths of steel.

The trucks, couplings, bumpers, etc., were exact duplicates of those on the muck-cars, so that these tool-cars could be placed anywhere in the trains.

Hollow drill-steel $1\frac{1}{8}$ in. in diameter, Bulldog brand, was

used; and the bits preferred were of the usual cruciform pattern.

To obtain and keep in such a high altitude, especially during the severe winter weather, the best class of men, it was necessary to build at both ends of the tunnel warm and comfortable quarters for the employees, as well as repair-shops, stores, and hospitals. Commodious buildings were erected, and, instead of crowding the men into the usual bunk-houses, the houses were divided up into rooms like a hotel, not more than two men being assigned to one bed-room. Each of these rooming-houses contained a general sitting-room, a bath-room, and a wash-room. As a further inducement to obtain and retain the best class of workmen, the dining-rooms were very comfortably furnished and run on a most generous plan. The men were charged $1 per day for board, but, even at this rate, expenditures exceeded receipts by over $1,000 during the construction of the tunnel; all of which was more than compensated for by the class of men retained and the good feeling engendered by comfortable quarters, good living, and liberal treatment.

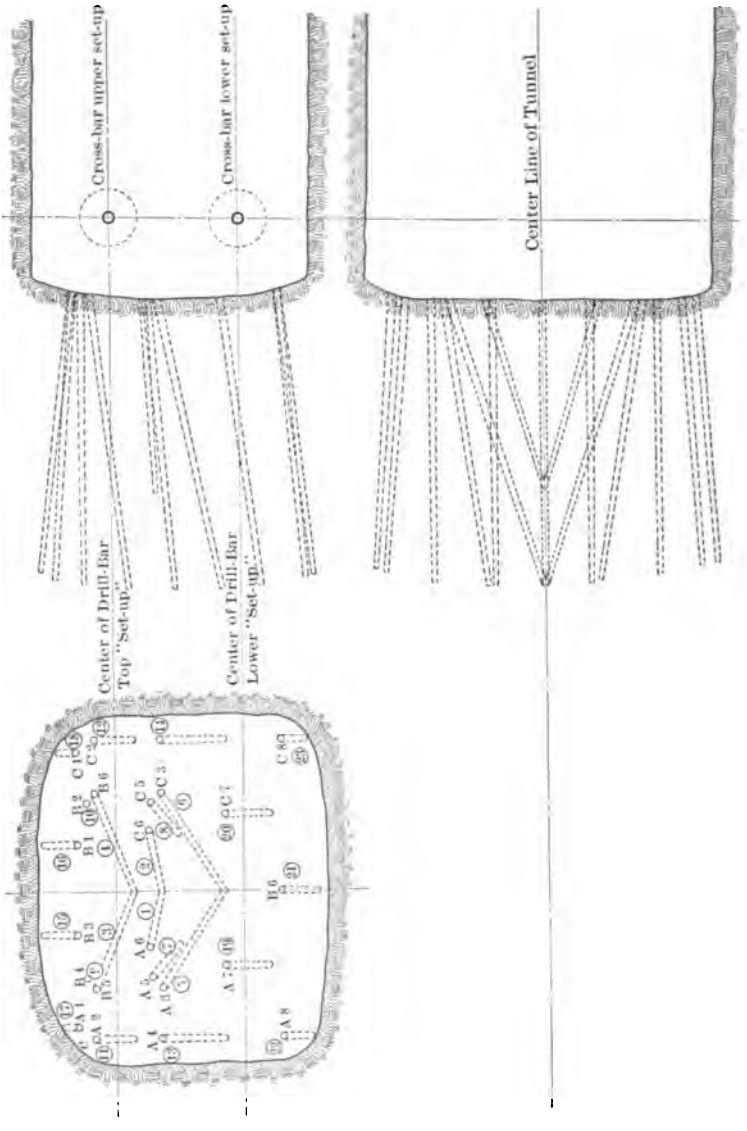
A store had to be maintained on each side of the mountain for the convenience of the employees; and, to prevent dissatisfaction from this cause, no article was sold at more than 10 per cent. profit. Liquor of any kind was not allowed in camp. Special care was taken to insure the safety of the employees, with the result that no one was killed and only one man very slightly injured during the whole progress of the work. Repair- and blacksmith-shops were erected at both ends of the tunnel, and, among other tools, each shop was supplied with a Leyner drill-sharpener.

The power-plant was completed and turned over to the contractor on Mar. 15, 1910, and was in continuous operation from that time until the completion of the tunnel, no repairs or shut-downs of any kind being necessary, except a few minutes stoppage at long intervals to tighten up driving-belts. The machinery was found to be entirely suitable and adequate to the service; and air-pressures of 120 lb. at the power-house were easily maintained when all the drills, blacksmith-forges, and sharpening-machines were in operation.

Ventilation was provided for by carrying a 16-in. air-pipe

FIG. 6.—TUNNEL-CAR USED IN CONSTRUCTION OF LARAMIE RIVER TUNNEL.

FIG. 7.—CAR FOR TRANSPORTING DRILLS, LARAMIE RIVER TUNNEL.



Cross-section shows position and order in which holes are drilled. Left-hand drill designated *A*; center, *B*; right, *C*. Figures in circles give order of firing.
Longitudinal section shows location of holes and position of cross-bar on upper and lower "set-ups."
Plan shows position and depth of holes. Cut holes are drilled to intersect to insure simultaneous explosions.

FIG. 8.—CROSS- AND LONGITUDINAL SECTIONS AND PLAN OF LARAMIE TUNNEL AT FACE.

to within 150 ft. of each face. This pipe was of No. 16 gauge sheet-steel, dipped in asphalt, and put together with the usual slip-joints, wrapped with tar-saturated canvas wherever an air-tight joint could not be made without it. Four-inch pipes were used for conveying air from the compressors into the tunnel, but for a few hundred feet from the face this size was reduced to 3- and 2-in., the smaller sizes being taken up and replaced by 4-in. as the tunnel advanced.

The water to supply the Leyner drills on the eastern heading was taken from a small stream above the tunnel and conveyed to the face by a 1-in. pipe. At the western end of the tunnel no provision of this kind was necessary, as the tunnel supplied not only the necessary water but, unfortunately, an over-abundant surplus.

In the early stages of the work numerous tests were made to determine the best and most economical explosive for use in the tunnel. The explosives tested varied in strength from blasting-gelatine, having the same strength as pure nitro-glycerine, to powders having only 50 per cent. of that strength. As the result of these experiments, it became clear that, except where the granite was extraordinarily hard and tough, Repauno 60-per cent. gelatine gave as good results as explosives of higher grade.

Insolid and Z. L. fuse, both of German manufacture, having a speed of about 1 ft. in 40 sec., were used, with 6x caps clamped to the fuse by a California "crimper." The fuse was cut to exactly 10-ft. lengths for all of the holes, except for the lifters, which were 2 ft. longer; the reason for the increased length of the lifter-fuse being that, since these holes could not be examined after the charge had been fired as readily as the holes in the face, it was desirable to have them explode some time after the other shots, so that the reports could be clearly heard and counted. Where the ground was wet, the fuse was tarred for 1 ft. back from the cap, the tar being worked in around the edges of the cap very carefully, and the remainder of the fuse to within 2 ft. of the end was thoroughly coated with heavy axle-grease. Where the heading was dry, the fuse was slit and ignited with candles; but where it was wet American fuse-lighters were used with excellent results.

quite easy to bring the bar, drills, etc., over the pile of muck, without waiting for it to be removed. Jacking the cross-bar into place seldom took more than from 6 to 8 min., and clamping the drills on the bar, attaching the hose, and starting them up consumed, according to conditions, anywhere from 5 to 15 minutes.

With the Leyner drills used there was, of course, no dust in the face, and the stream of mixed air and water passing through the hollow steel not only kept the bit cool, but removed the cuttings as fast as they were formed, thereby making it unnecessary to stop the drills and take out the steel for the purpose of cleaning out the holes. Unless the rock was unusually hard, each drill could be run down the full length of the step between the different sizes, which was usually maintained at 18 in. on the starters and 24 in. on the long drills, without any necessity for change.

The directions of the holes and the order in which they were drilled are shown in Fig. 8.

The time required to complete the 18 holes which were drilled from the top set-up ranged from 3 to 4.25 hr., the holes averaging about 7 ft. 6 in. in depth. The time required to lower the bar from the upper to the lower set-up varied from 15 to 25 min., and the three lifters and two relief-holes were usually run down in less than an hour. Disconnecting the hose, removing the drills and taking down the cross-bar, and carrying them back to a place of safety in the tunnel, consumed from 15 to 20 minutes.

Blowing out the holes, loading, and splitting generally occupied from 20 to 30 minutes.

Loading the holes, the most critical operation in the cycle, was performed by two machine-men and two helpers—the foreman directly superintending the work and deciding on the proper amount of powder to be used in each hole. A stick of 60-per cent. powder was placed in the bottom of each hole below the primer, and on top of the primer there were usually placed five sticks of powder, except in the cut-holes, each of which received three or four sticks extra. The three lifters were loaded to the collar, the additional amount of explosive being used for the purpose of throwing the muck as far back from the face as possible.

The fuses were always ignited and the charges fired in rotation, the following order being carefully observed:

1.	Short cut-holes,	Nos. 1 and 2.
2.	Top cut-holes,	Nos. 3 and 4.
3.	Lower cut-holes,	Nos. 5 and 6.
4.	Relief cut-holes,	Nos. 7 and 8.
5.	Relief back-holes,	Nos. 9 and 10.
6.	Top side-holes,	Nos. 11 and 12.
7.	Lower side-holes,	Nos. 13 and 14.
8.	Back-holes,	Nos. 15 and 16.
9.	Corner back-holes,	Nos. 17 and 18.
10.	Lower relief-holes,	Nos. 19 and 20.
11.	Center lifter,	No. 21.
12.	Left-hand lifter,	No. 22.
13.	Right-hand lifter,	No. 23.

The usual practice of tamping the holes over the explosive was soon discontinued, as it was found that with such heavy charges the powder formed its own tamping, with the further advantage that when the holes were loaded to the collar the rock was more thoroughly pulverized and consequently much easier shoveled into the cars than when lighter charges were used.

Holes will occasionally miss fire, even when loaded with the greatest care; and when no tamping is employed they can be afterwards fired by simply pushing a primer down tightly upon the unexploded charge, without taking the risk of performing that most dangerous of all operations, picking the tamping out of a "missed" hole.

At first, each pair of fuses was lighted about 10 sec. before the next—which, on 40-sec. fuse, gave 3-in. steps on the receding line of fire. This interval, however, being repeated on 10 pairs of holes, occupied considerable time, and the smoke became so intolerable that some method of expediting the rate of fuse-lighting had to be adopted. The one which proved most satisfactory was exceedingly simple. The foreman cut 22 in. from the ends of the fuse protruding from the short cut-holes; 20 in. off the fuse from the upper cut-holes; 18 in. off the fuse in the lower cut-holes, and so on. This automatically provided a difference of 2 in. in the distance the fire had to travel, and, even when the fuse-ends were lighted as rapidly as possible, at least another inch was represented by

the time between the lightings, so that the two shot-firers could secure the necessary interval between the explosions, and yet get away from the face before the smoke from the burning fuse became too dense.

The following tabular recapitulation of the drilling-operations shows that the men could not only complete a round in an 8-hr. shift, but had sufficient extra time to provide for shooting missed holes or taking care of any of the minor difficulties which often arise in tunnel-work:

Periods Occupied in Various Operations.

Exhausting smoke from face,	10 to	12 min.
Picking down roof and sides,	5 to	10 min.
Jacking cross-bar in place,	6 to	8 min.
Attaching drills, making hose- and water-connections,	5 to	15 min.
Drilling from top set-up,	3 hr. to	4 hr. 15 min.
Dropping horizontal bar to lower position,	15 to	20 min.
Drilling on lower set-up,	1 hr. to	1 hr. 15 min.
Removing drills, cross-bar, hose, etc.,	15 to	20 min.
Blowing out holes, loading and firing,	20 to	25 min.
Ignition to explosion of last hole,	8 to	8 min.
Total time required to complete cycle of operations,	5 h. 24 min. to	7 h. 28 min.

Muck from the face was hauled out to the dump in 10-car trains, each car having a capacity of 16.5 cu. ft. A single track of 18-in. gauge, 16-lb. rails, was used for the entire distance, with a passing-switch, located about one mile from the eastern portal. Each train was hauled by two mules, driven tandem.

The empty cars, as they were brought in, were drawn up as close to the loaded train in the face as possible; then they were "stepped" off the track and thrown over on their sides, and the mules were attached to the loaded train, and started for the dump. As each car was fitted with a brake provided with a long handle and ratchet-bar, it was possible to graduate the brake-resistance so that the train could be easily and safely handled on the 1.7-per cent. grade.

As soon as the loaded train had started from the face, two of the empty cars were righted, placed on the track, and pushed up as far into the muck-pile as the track could be cleaned, and the rear car of the two was "stepped" off the

track and thrown over on its side. When the front car had been loaded it was shoved back to a point just beyond the farthest empty car, and the trammers, returning, uprighted and brought back with them empty car No. 3, which was pushed up to the car in the face, which, in the meantime, had been righted and loaded; and then No. 3 was "stepped" off and thrown over on its side, to permit loaded car No. 2 to pass out, when No. 3 was instantly righted and placed on the track, and the process of filling it carried on while the trammers were running out car No. 2 and bringing back car No. 4 to the face. This operation was repeated until the entire train of empties had been uprighted and loaded. By this time another train of empties would have arrived from the dump, and the process was continued until the face of the tunnel was cleaned. The number of car-loads broken by a round of holes ranged from 50 to 60.

By reason of the heavy charges used, the rock was thoroughly pulverized and but little picking was required, and, as the floor of the tunnel was kept covered with steel plates ⅜ in. thick, 3.5 ft. wide, and 7.5 ft. long, shoveling was easily and expeditiously performed with No. 5, D-handled, square-pointed shovels.

Four men were employed in filling, and two in uprighting and tramming the cars. By selecting good men and teaching them how to "muck," it was possible to get the broken rock removed, the tunnel cleaned up, track laid, and the steel plates moved up to within 3 ft. of the new face, by the time the drillers were ready to fire.

Conditions were usually extremely favorable for rapid work; but at times shear-zones were encountered in the granite, where the rock was softened to such an extent that the tunnel had to be timbered. This, of course, put heavy firing out of the question and materially reduced the rate of progress. In May, 1911, the men had great hopes of making over 700 ft. in the month's run. From the 1st to the 25th, inclusive, they had driven 568 ft., when they ran into a softened shear-zone which retarded their progress to such an extent that they drove only 67 ft. during the six remaining days of the month.

From Mar. 1 to Mar. 8, 1911, inclusive, the tunnel was driven, in a single heading, 192 ft., a daily average of 24 ft.; but the highest rate of progress was made during the last four

days of January, 1911, when the tunnel was driven 112 ft., or 28 ft. per day.

The amount driven each month on the tunnel at both portals is given below, the record month being March, 1911, with a distance of 653 feet:

Monthly Record of Driving Laramie Tunnel.

	East Portal.	West Portal.
1910.		
January,	302	
February,	315	
March,	350	202
April,	354	279
May,	513	336
June,	429	388
July,	443	371
August,	527	293
September,	485	286
October,	420	28
November,	424	
December,	482	
1911.		
January,	609	
February,	420	
March,	653	
April,	583	
May,	635	
June,	576	
July,	497	
August,	106	
Totals,	9,123	2,183
Grand total,	11,306 feet.	

Work on the west heading was discontinued Oct. 3, 1910, because the up-grade haulage of muck, hoisting it through the incline, and pumping out the water which came in from all sides, combined to make the cost per foot much heavier than in the more favorably situated heading from the east portal, where the 1.7-per cent. down-grade gave perfect drainage and greatly cheapened transportation from the face. By this date, moreover, sufficient work had been done to demonstrate that the tunnel could be completed from a single heading within the specified time, so that no risk was incurred by the contractor in restricting operations to the more accessible and more cheaply operated east portal.

During March, April, and May, 1911, the record for distance driven, drilling, powder consumed, and cars of waste sent out was:

1911.	Feet of Completed Tunnel.	Number of Holes Drilled.	Linear Ft. of Holes Drilled.	Avg. Linear Ft. of Holes Drilled Daily.	Sticks of Powder Used.	Cars of Muck Sent Out.
March,	653	1,965	14,330	154	14,808	4,983
April,	583	1,759	12,510	139	16,171	4,765
May,	635	1,985	15,263	164	18,311	5,156

Considering the hardness of the rock, the speed attained in drilling, as shown by the figures above, was exceptionally good; but even these averages fall considerably below what was possible with the equipment used. For instance, a number of the best drill-runners were able to average over 60 ft. of holes per shift, one of them making a monthly shift-average of 61.68 ft.; another of 61.75 ft.; and a third of 61.86 feet.

While this work shows a great advance over current American practice, it still falls behind the records obtained in the best examples of European tunnel-driving. A direct comparison, however, is not quite fair to the United States, since the Alpine tunnels are very much longer than anything yet attempted in this country. At first sight, it would seem that additional length would tend to retard instead of accelerate the rate of progress; but this is not the case. It has been clearly shown that the increased length of transportation and difficulty of ventilation are much more than offset by the improved conditions and the perfection of organization effected by time and experience. As a rule, the greater the magnitude of the undertaking the more thorough the preparation; and the time and labor expended in studying conditions and designing plants for the different Alpine tunnels have been more than justified by the results obtained. European tunnel-engineers have also the advantage of being able to select their employees from an almost unlimited supply of highly-skilled workmen from the Tyrol, Switzerland, and Piedmont, which gives them an incomparably better selection than can be drawn from our heterogeneous labor-supply.

It is, however, confidently believed that, under favorable conditions, with tunnels of sufficient length, thoroughly up-to-date plants, and well-selected crews, backed by careful study

and vigorous management, we will soon be able in this country to equal, or perhaps even surpass, the best European records.

In conclusion, I wish to acknowledge my indebtedness to John A. Davis, of the Bureau of Mines; Charles Hedke, Chief Engineer; B. S. Coy, Resident Engineer; and James A. McIlwee, contractor, for valuable assistance rendered in the collection of data for this paper.

A Modification of the "Gay Lussac" Method for Silver-Bullion Containing Tin.*

BY LUIS EMLYNN SALAS, NEW YORK, N. Y.

(San Francisco Meeting, October, 1911.)

If the ordinary wet method be attempted for silver-bullion containing tin, much trouble is experienced, varying with the amount of tin present. Even with a percentage as low as 0.05, the end-point is masked by a persistent turbidity, while with amounts ranging from 0.5 per cent. upward, the determination of the exact end-point is impossible, owing to the finely divided or colloidal metastannic acid resisting all efforts to cause it to settle, or to give a clear supernatant liquid.

The object of the following experiments was to seek a remedy for these difficulties, and to find the conditions whereby the Gay Lussac method might be applied directly to these bullions. Material containing tin is met with frequently in bullion from Mexico or Bolivia; and tin is carried (sometimes in large quantities) in jewelers' sweeps.

1. A bullion was prepared containing Ag, 74; Cu, 25; and Sn, 1 per cent.

A weighed amount was treated by the humid assay-method as if it were an ordinary bullion free from tin. Ten cc. of HNO_3 (1.21) was added, and the bottle was heated until all traces of

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red fumes had been expelled. Before heating, the solutions were moderately turbid; and after heating, the turbidity greatly increased.

One hundred cc. of normal NaCl solution was added and the bottle was shaken vigorously at intervals during 30 min. The turbidity persisted. The bottle was allowed to stand over night, and the next day it was found to be only moderately clear.

A slight agitation reproduced the troublesome turbidity, the precipitate failing to clear during the afternoon.

2. Various attempts were made to cause the precipitate to settle.

(*a*) The bottles were placed in a centrifugal machine and whirled for several minutes, but no appreciable settling took place.

(*b*) Mixtures of ether and alcohol were added, and the bottle was agitated. No desirable result was obtained.

(*c*) Several other assays were weighed out and treated as above with similar results. A small amount of egg albumen, in water solution, was added and heat applied to coagulate the albumen. Little or no clearing took place after shaking and allowing to stand. Frothing gave trouble, and the idea was abandoned.

(*d*) The production of heavy precipitates, such as $BaSO_4$, etc., to carry down the tin failed to give good results. Other attempts were made to cure the trouble, but none gave satisfaction.

It became evident that to seek a "prevention" would be more likely to lead to success.

3. Trials were made with many substances to keep the tin in solution, and the best results were obtained with the organic acids, tartaric and citric; but oxalic acid failed.

Two rows, *A* and *B*, of nine test-tubes each were arranged, and into each tube was placed a piece of tin weighing approximately 50 milligrams.

To 1, 2, 3 of set *A* was added . . .	0.1 g. of tartaric acid.
To 4, 5, 6 of set *A* was added . . .	0.25 g. of tartaric acid.
To 7, 8, 9 of set *A* was added . . .	0.5 g. of tartaric acid.
To 1, 2, 3 of set *B* was added . . .	0.1 g. of citric acid
To 4, 5, 6 of set *B* was added . . .	0.25 g. of citric acid.
To 7, 8, 9 of set *B* was added . . .	0.5 g. of citric acid.

Five cc. of HNO_3 (1.21) was added to each of the 18 tubes and in every case the contents were warmed.

To check the above results, three tests were made without the addition of an organic acid. The liquid remained clear where either tartaric or citric acid was used; whereas, those tests containing neither tartaric nor citric acid became turbid on solution in the cold, and the turbidity increased on heating.

Even after considerable heating the tests Nos. 4 to 9 in each set remained clear.

But little difference could be noticed between the effects of tartaric and citric acids; if any, tartaric acid gave a slightly more satisfactory solution.

4. Further tests with tartaric acid were then made. A bullion containing Ag, 74; Sn, 2; and Cu, 24 per cent. was prepared, and three convenient quantities (*A*, *B*, and *C*) were taken for assaying. To each lot was added a small amount of tartaric acid (approximately 0.5 g.), and then 10 cc. of HNO_3 (sp. gr. 1.21). *C* alone gave a solution sufficiently clear for titration. (It was noted that *C* had not been heated as strongly as *A* or *B*.)

Two other tests, *D* and *E*, were made, 1 g. of tartaric acid being added to each, the bullion and crystal of acid being dissolved in HNO_3, as before. *D* became turbid on heating. It was observed that the turbidity greatly increased just before the red fumes disappeared. Test *E* also became turbid even in the cold. The precipitate appeared "silky" on rotating the bottle, and on warming to about 60° C. it diminished considerably, the solution becoming almost clear. Titration was attempted, but on diluting the silver solution with the 100 cc. of NaCl solution, the liquid would not clarify, even after prolonged agitation. The titration to the end-point was impossible with this assay.

5. It was observed that with regard to turbidity, the results obtained depended upon the heat-treatment. Therefore another series of experiments was undertaken upon ordinary bullion free from tin, with the object of testing the effects produced with and without tartaric acid, and with and without the application of heat. It was found that the presence of the tartaric acid and the omission to remove the red fumes (which removal is generally recommended and sometimes insisted

upon) had no observable effect upon the accuracy of the assay. This last point being contrary to general practice, additional experiments were made to confirm it, giving the following results:

Assay.	Silver (c. p.) Taken. Milligrams.	Silver Found. Milligrams.
No. 1,	1005.55	1005.58
No. 2,	1005.81	1006.08
No. 3,	1005.49	1005.58
No. 4,	1005.71	1005.58

In all of these assays, c.p. silver was dissolved in HNO_3 (1 : 1) in the cold and titrated; all with tartaric acid; no heat applied.

6. A solution of NaCl was prepared and standardized, the standard equaling 1004.33 mg. of silver.

This solution was checked, and gave the following results:

Assay.	Amount of c. p. Ag Taken. Milligrams.	Silver Found, Using the Standard Mentioned Above. Milligrams.
No. 1,	1005.44	1005.58
No. 2,	1005.33	1005.08
No. 3,	1005.27	1005.33
No. 4,	1006.63	1006.83
No. 5,	1005.92	1005.83
No. 6,	1005.22	1005.08

These tests were finished with NaCl.

To assays Nos. 5 and 6 there was added 2.5 cc. excess of decime NaCl, and the solutions were titrated back with decime $AgNO_3$, in the endeavor to ascertain the difference in end-point, finishing with NaCl and with $AgNO_3$. The solutions were agitated vigorously, but did not clear readily, and the end-point was obscure. This confirms the effect noticed by Aaron.

7. In the calculations the decime solution, as noted, was added 0.5 cc. at a time, and one-half of the last addition that gave a precipitate was counted.

Thus, 0.5 cc.
0.5 cc.
0.5 cc.
0.5 cc., no precipitate, equals 1.25 cc.

In order to determine if the addition of the decime solution in smaller amounts would make the end-point more exact, two tests were made, using 0.2 cc. decime NaCl.

A. 1005.21 mg. Ag. NaCl, 100 cc.	*B*. 1005.54 mg. Ag. NaCl, 100 cc.
+ 0.2 + 0.2 + 0.2 + 0.2 + 0.2.	+ 0.2 + 0.2 + 0.2 + 0.2 + 0.2 + 0.2.
No precipitate.	No precipitate.
= 1004.33 + 0.7.	= 1004.33 + 0.9.
= 1005.03 Ag found.	= 1005.23 Ag found.

The end-point was possibly less sharp, and the results were not improved.

8. In some of the previous assays certain effects were thought to be due to the amount of free HNO_3 present.

Two tests, *A* and *B*, were made to determine this point.

The c. p. silver was in each case dissolved in 10 cc. of HNO_3 without tartaric acid.

A was evaporated to incipient dryness. *B* was evaporated so that, on cooling, crystals of $AgNO_3$ separated. To each assay 100 cc. of NaCl solution was added, and the bottles briskly agitated. Neither of these solutions cleared well. *A* was especially cloudy. It was re-agitated and allowed to stand for 15 min., and titration continued, giving the following results:

A. Amount of silver taken, 1005.92 mg.; amount found, 1005.58 mg.
B. Amount of silver taken, 1005.50 mg.; amount found, 1005.83 mg.

Excess of $AgNO_3$ was then added to each test, but the solution did not clear well even after prolonged agitation, *A* being especially troublesome. To each, 5 cc. of HNO_3 (1.21) was added, and the agitation was repeated. Both solutions cleared perceptibly. The clearance was not complete in *A*, but *B* was very good. It appears, therefore, that the assay is best conducted in such a manner that considerable free HNO_3 remains after solution of the silver has taken place, and that excessive evaporation is disadvantageous even with tin-free bullions.

9. To determine the best strength of HNO_3, the following experiments were made:

About 250 mg. of bullion *D* (similar to *C*) was added to each of six test-tubes.

No.	H_2O Added. Cubic Centimeters.	HNO_3 (1.42) Added. Cubic Centimeters.	Results.
1	10	2	Few bubbles, dissolved after 5 hr.
2	8	4	Similar.
3	6	6	Quick action, clear solution.
4	4	8	Quick action, white sediment.
5	2	10	Quick action, white sediment.
6	0	12	No action at first. Ultimately heavy white sediment.

From the above tests, it was concluded that (1 : 1) HNO_3 (ordinary strong parting-acid) gives the best results.

The results obtained with bullion *D* were:

Amount Weighed Out. Milligrams.	Tartaric Acid. Grams.	NaCl. 100 cc.	Ag Equivalent. Milligrams.	Fineness.
1339.34	2	+ 0.75 dec.	1005.08	750.4
1339.54	2	+ 0.75	1005.08	750.3
1339.61	2	+ 0.75	1005.08	750.2
1340.65	2	+ 2.0	1006.33	750.6

10. It was found desirable, in assaying silver bullion containing tin, that the tartaric acid should be immediately available for the dissolution of the tin. We therefore proposed to dissolve the 2 g. of tartaric acid (a convenient quantity for bullions containing about 5 per cent. of Sn) in about 3 cc. of water in the assay-bottle before adding the HNO_3.

11. Bullion *A* was prepared containing 0.1 per cent. of tin; fineness, 750. The copper used was found to contain a little silver (this was discovered later when looking for the reason of the high results).

No.	Amount of Bullion. Milligrams.	Tartaric Acid. Grams.	NaCl. 100 cc.	Ag Equivalent. Milligrams.	Fineness.
1	1338.76	1	+ 0.75 dec.	1005.08	750.7
2	1338.91	0.5	+ 0.75 dec.	1005.08	750.6
3	1338.84	1	+ 1.0 dec.	1005.33	750.8
4	1338.53	0.5	+ 1.0 dec.	1005.33	750.9

A cloudiness appeared in the above tests when bringing the silver into solution, but this cloud disappeared on warming. No. 3 was actually brought to the boiling-point without special trouble occurring, the percentage of tin being small. But this heating is not permissible with bullions containing large amounts of tin.

Additional experiments, Nos. 5 and 6, were carried out as follows: No. 5 contained no tartaric acid, and the solution was heated till the red fumes were expelled. The solution did not clear well, and the end-point could not be ascertained with certainty.

No. 6 contained 1 g. of tartaric acid, and was dissolved at room-temperature. The solution remained clear till the end, and gave a sharp end-point. The fineness found was 750.8.

12. Bullion *B* contained 1 per cent. of Sn, and was approximately 750 fine.

The results obtained were similar to those of Bullion *A* in experiment 11, except that the turbidity, where occurring, due to lack of tartaric acid, was more marked.

Bullion *C* contained 5 per cent. of Sn, and was approximately 750 fine. In accordance with the observation made in experiment 6, with regard to the better clearing of the liquid when finishing with NaCl rather than with $AgNO_3$, in all these assays an amount of bullion was weighed out, so that excess of silver remained in the assay-bottle after adding the 100 cc. of NaCl. Citric and tartaric acid were tried. The assay containing citric acid would not settle at all, the liquid retaining its milky appearance for a whole day; moreover, the AgCl did not granulate well. Other tests confirmed the fact that citric acid is useless for the purpose. Tartaric acid, however, gave exceedingly good results so far as the clearing of the solution was concerned.

The experiments with this bullion containing 5 per cent. of Sn seemed to indicate:

(1) That the best clearance was obtained when few additions of decime NaCl were necessary.

(2) That the bullion must be dissolved in the cold. (In some cases the bottles were cooled under the tap with advantage, after dissolving the tartaric acid, and before adding the HNO_3).

(3) That 2 g. of tartaric acid gave satisfactory results; 1 g. appeared insufficient, and 3 g. indicated some interference with the end-point.

Above all, the best results were obtained when sufficient bullion was taken to provide an excess of silver, the titration taking place entirely on the salt side.

13. Bullion *E*, made up of similar composition to that of *D* (5 per cent. of Sn), was granulated, and, after granulation, was found to be thickly coated with oxides. Bullions that are oxidized to a considerable extent give trouble in dissolving, as heat, necessary for complete dissolution, causes the separation of metastannic acid. Moreover, it was observed that although a clear solution (strongly acid) might sometimes be obtained, an intense turbidity was produced on diluting with the 100-cc. NaCl solution, and that settling was impossible even after prolonged agitation, due to the alteration of the state of hydration of the tin oxide.

It is to be noted here that all bullions that had been heated strongly gave precipitates (on dilution) which did not settle.

14. Bullion *F*, containing approximately Ag, 75; Cu, 20; and Sn, 5 per cent., was poured into the form of a conical button, which was flattened and rolled, thus increasing the chances of obtaining a uniform assay-piece.

The bullion was first placed in the bottle, then 2 g. of crystals of tartaric acid, then 3 or 4 cc. of distilled water was added, and the liquid heated till the crystals had dissolved. The solution was well cooled, and then was added 10 cc. of HNO_3 of a strength slightly greater than (1 : 1) to counterbalance the water added to dissolve the tartaric acid.

The bullion was allowed to dissolve in the cold, and complete solution took place in about 20 min. The following results were obtained:

No.	Bullion Taken. Milligrams.	NaCl. 100 cc.	Ag Equivalent. Milligrams.	Fineness.
1	1338.60	+ 1.75 dec.	1006.08	751.5
2	1338.74	+ 1.75 dec.	1006.08	751.5
3	1338.13	+ 1.50 dec.	1005.83	751.3
4	1338.21	+ 1.25 dec.	1005.58	751.4
5	1338.49	+ 1.0 dec.	1005.33	751.1
6	1337.54	+ 0.25 dec.	1004.58	751.0
7	1337.60	+ 0.25 dec.	1004.58	751.0
8	1337.48	+ 0.25 dec.	1004.58	751.1
9	1337.64	+ 0.25 dec.	1004.58	751.0
10	1337.79	+ 0.25 dec.	1004.58	750.9
11	1338.58	+ 1.25 dec.	1005.58	751.2
12	1337.83	+ 0.25 dec.	1004.58	750.9
			Average,	751.16

The above results are very concordant, but there was a possibility of a uniform error existing. To check this point known weights of Ag (c.p.), Cu, and Sn were taken and placed loosely in assay-bottles, *A* and *B*.

A contained Ag (c. p.), 1005.66; Cu, 250.80; and Sn, 60.46 mg.

B contained Ag, 1005.52; Cu, 250.44, and Sn, 77.66 mg.

These assays were treated by the method given in the *résumé* at the end of this paper.

Amount of NaCl required in A = 100 cc. NaCl + 1.50 dec.
B = 100 cc. NaCl + 1.25 dec.

Ag equivalent of A = 1005.83.

Ag equivalent of B = 1005.58.

The above method of treatment is the proper one for bullions containing tin.

15. An attempt was made to apply the method given in Experiment 14 directly to a 20-per cent. Sn alloy.

The tests were prepared thus:

	Ag (c. p.). Milligrams.	Cu. Milligrams.	Sn. Milligrams.
Bottle *A* contained	1005.20	250.25	200.15
Bottle *B* contained	1005.52	250.61	200.30

The assays were treated by the method given in the *résumé*. Good clear solutions were obtained while the HNO_3 remained relatively concentrated; but on dilution with the 100-cc. NaCl solution a heavy cloud was produced, which did not clear by agitation and standing. If the percentage of tin in the bullion is so great that it interferes with this method, decrease the amount of bullion taken and make up the deficit of silver by adding c. p. Ag.

16. As a check upon the method it was desirable to make a fire-assay.

Bullion *F* having been exhausted, another bullion, *G*, was prepared of the following composition:

Ag, 22.5; Cu, 5; and Sn, 1.5 g., having a fineness of 750, and containing 5 per cent. of tin.

The strong NaCl solution was re-standardized. The standard had not altered (= 1004.33 mg. of Ag). The bullion was treated by the method indicated in the *résumé*. The results were:

	Bullion. Milligrams.	NaCl 100 cc.	Ag Equivalent. Milligrams.	Fineness.
A.	1339.66	+ 0.25 cc. dec.	1004.58	749.9
	1339.78	+ 0.25 cc. dec.	1004.58	749.9
	1340.14	+ 0.75 cc. dec.	1005.08	750.0
	1341.04	+ 1.25 cc. dec.	1005.58	749.8
	1340.21	+ 1.25 cc. dec.	1005.58	750.3
	1341.47	+ 1.75 cc. dec.	1006.08	750.0
			Average,	749.98

The bullion-assays marked *A* were titrated by an independent assayer, the amounts of bullion taken being "unknown."

Other results obtained by independent assayer upon unknown

quantities of alloy were: 749.9; 750.3; 749.8; 750.1; 750.4; averaging 750.1.

17. To check the results of experiment 16, the following fire-assays were made:

Three bullion assay-pieces were weighed out:

(1) 500.05 mg. (2) 500.44 mg. (3) 500.19 mg.

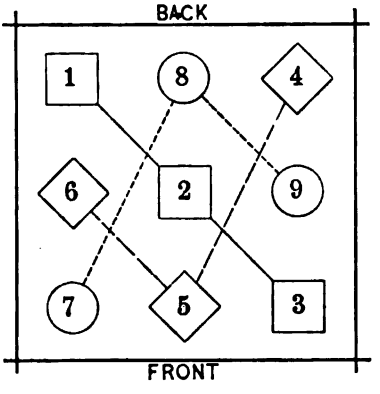
Three checks were made up containing tin:

(4) 375.12 mg. Ag.	(5) 375.26 mg. Ag.	(6) 375.12 mg. Ag.
100.10 mg. Cu.	100.12 mg. Cu.	100.08 mg. Cu.
24.95 mg. Sn.	25.16 mg. Sn.	25.06 mg. Sn.

Three checks containing no tin (these were run in order to determine if the tin had any effect upon the results for silver):

(7) 375.30 mg. Ag.	(8) 375.00 mg. Ag.	(9) 375.00 mg. Ag.
100.42 mg. Cu.	100.20 mg. Cu.	100.34 mg. Cu.

The nine cupels were surrounded on all sides by old cupels, and a bar of fire-brick was placed at the door of the furnace. None of the beads sprouted. The relative positions of the bullions and checks were as shown in diagram:



Weight of prills obtained:

(1) 368.39 mg.	(8) 368.60 mg.	(4) 368.90 mg.
(6) 368.88 mg.	(2) 369.70 mg.	(9) 369.02 mg.
(7) 370.32 mg.	(5) 368.52 mg.	(3) 370.07 mg.

Loss on (8) = 6.40 mg.; loss on (4) = 6.22 mg.

Loss on (6) = 6.42 mg.; loss on (9) = 5.98 mg.

Loss on (7) = 4.98 mg.; loss on (5) = 6.74 mg.

Mean loss on back row = 6.31 mg. (add to assay 1).

Mean loss on middle row = 6.11 mg. (add to assay 2).

Mean loss on front row = 5.86 mg. (add to assay 3).

Therefore,

368.39 + 6.31 = 374.70 mg. corrected Ag found in (1).
369.70 + 6.11 = 375.81 mg. corrected Ag found in (2).
370.07 + 5.86 = 375.93 mg. corrected Ag found in (3).

Fineness as determined by (1) = 749.3, (2) = 750.9, (3) = 751.1 } Average, 750.4

Since the checks 7, 8, 9 contained no tin, they are not strictly comparable with the bullion-assays 1, 2, 3.

Calculating from 4, 5, 6, containing tin, the results are:

Prill No. 1: 368.39 + loss on 4 (6.22 mg.) = 374.61 (= 749.1 fine).

Prill No. 2: 369.70 + loss on 6 (6.42 mg.) = 376.12 (= 751.1 fine).

Prill No. 3: 370.07 + loss on 5 (6.74 mg.) = 376.81 (= 753.3 fine).

Other checked fire-assays were run on the same bullion. Amounts weighed out: *A*, 500.05; *B*, 500.13; *C*, 500.12.

Prill from *A* (367.46 mg.) + loss on check (7.01 mg.) = 374.47 mg. (= 748.9 fine).

Prill from *B* (366.26 mg.) + loss on check (8.17 mg.) = 374.43 mg. (= 748.7 fine).

Prill from *C* (367.60 mg.) + loss on check (7.54 mg.) = 375.14 mg. (= 750.1 fine).

The average fineness calculated from these six fire-assays was 750.2.

NOTE.—The tests without tin were run as checks upon those containing tin. If the abnormal results obtained in 5 and 7 be discarded, the average losses on the assays with and without tin agree closely, indicating that the presence of tin has little or no effect upon the results of the fire-assay.

RÉSUMÉ OF METHOD.

1. Place a weighed amount of bullion in assay-bottle and add tartaric acid crystals. Then add 3 or 4 cc. of distilled water. Apply heat until the tartaric acid is dissolved.
2. About 2 g. of tartaric acid is a convenient quantity to use with 1 g. of a bullion containing up to 5 per cent. of tin. After dissolution of the tartaric acid, cool thoroughly.

3. 10 cc. of HNO_3 (1 : 1, or a little stronger) is now added, and the bottle rotated slightly to mix the two acids thoroughly. Keep the solution cool.
4. The amount of bullion to be taken must be such as to leave an excess of silver in the solution after the 100 cc. of NaCl has been added.
5. The bullion must be dissolved in the cold. It is not necessary to boil off the nitrous fumes.
6. The assay must not be heated after the HNO_3 has been added.

The Black Mountain Coal-District, Kentucky.

BY J. B. DILWORTH, PHILADELPHIA, PA.

(San Francisco Meeting, October, 1911.)

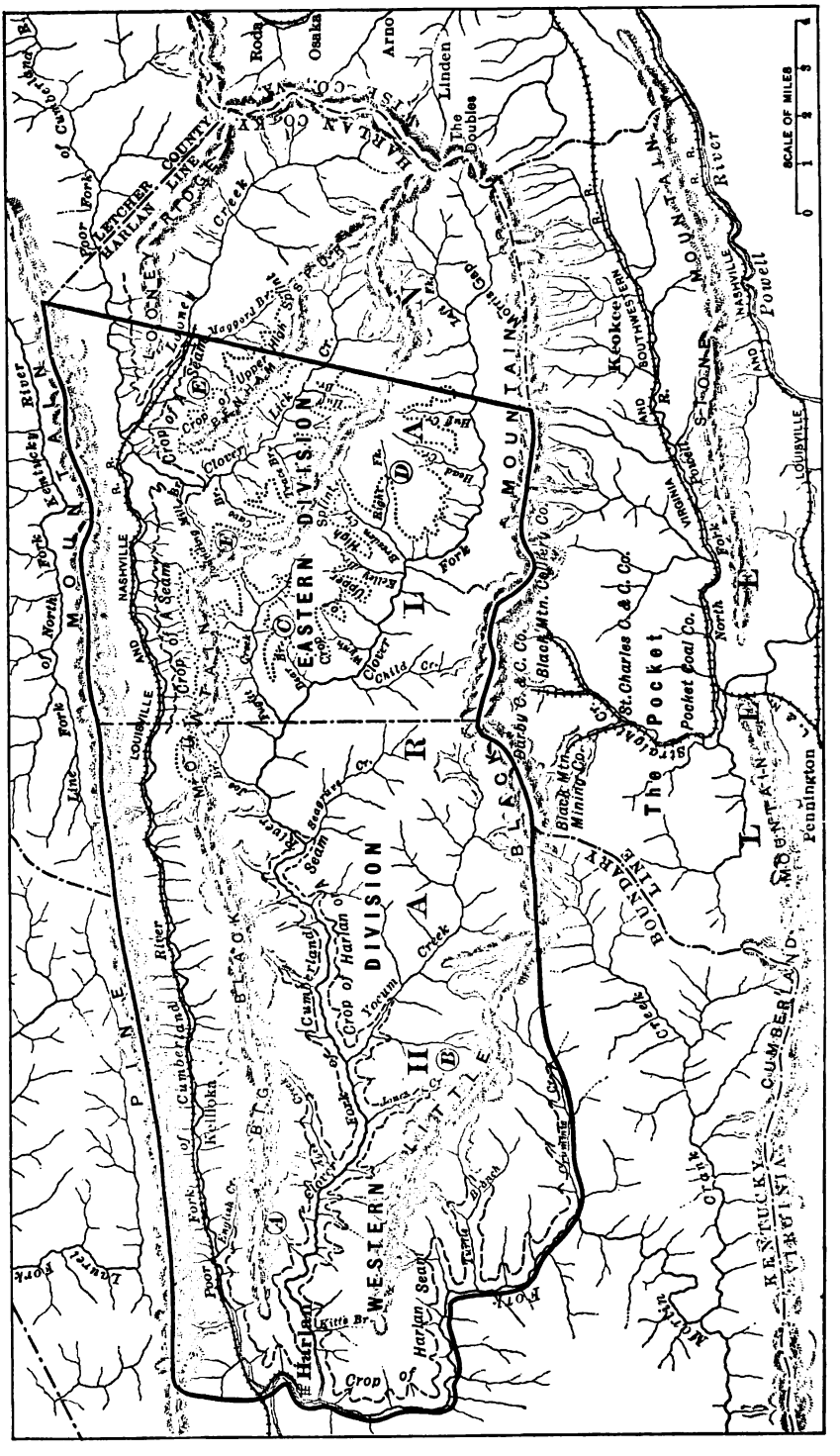
I. Introduction.

The purpose of this paper is, first, to give a general account of a little-known coal-district of SE. Kentucky, its topography, drainage, and mineral resources, for those who may be interested in its economic development; and secondly, so to describe the geologic features of the field as to furnish available data to those engaged on the broader problems of Appalachian coal-geology.

The data from which the major portion of the paper has been prepared were obtained, in professional investigations for private individuals and corporations, within the last decade.

The area under review, defined by a heavy line on the map, Fig. 1, lies between Little Black and Pine mountains, and extends from Harlan on the west to a N–S. line crossing Looney creek 3 miles from its mouth on the east. This area can be divided for description into a Western and an Eastern Division.

The character of investigation pursued in the Western Division, below Fugitt creek, was very different from that employed in the Eastern. In the former, only a rapid reconnoissance was attempted: such coal-openings as had been made by prospectors or by citizens in obtaining fuel were visited, their


SCALE OF MILES
0 1 2 3 4
LETCHER COUNTY
HARLAN LINE
Rock
Osaka
Arno
Linden
The Doubles
Powell
River
Kentucky
Cumberland
Poor Fork
Line Fork
North Fork
Crop of A Seam
Crop of Harlan Seam
EASTERN DIVISION
WESTERN DIVISION
BLACK MOUNTAIN
PINE MOUNTAIN
LITTLE MOUNTAIN
CUMBERLAND MOUNTAIN
KENTUCKY
VIRGINIA
Harlan
Pennington
Pocket
Pocket Coal Co.
St. Charles C. & C. Co.
Black Mtn. Coal Co.
Black Mtn. Mining Co.
Barbo C. & C. Co.
BOUNDARY LINE
Louisville and Nashville
Kentucky and Southwestern
Yocum Creek
Clover Lick
Child Cr.
Kitts Br.
Lone Cr.
Tenn. Branch
Sparkle Cr.
Hoot Cr.
Martin Cr.
Cumberland River
Pine Mountain
Little Mountain
Cumberland Mountain
Black Mountain
Harlan Seam
Crop of Harlan Seam
Crop of A Seam
Crop of Cumberland Seam
Crop of Upper Elkhorn Seam
Crop of Clover Lick Seam
Crop of Eastern Division Seam
Crop of Black Mtn. Seam
Crop of Pocket Seam
Crop of Eastern Division Seam
Crop of Harlan Seam
Crop of Cumberland Seam
Crop of Harlan Seam
Crop of Harlan Seam
Crop of Harlan

sections measured and, in many instances, sampled. Elevations were obtained barometrically and approximate locations sketched on a small-scale map.

In a field where the coals are so well opened, the reconnoissance method gives a very fair idea of the number and character of the principal beds and their relative positions, and affords much evidence for their correlation over wide areas, without permitting the more refined work required to discover all the seams of the measures and to determine the details of geologic structure.

In the Eastern Division, the territory is largely owned in fee simple by one large corporation, which had it thoroughly prospected prior to the inauguration of mining-operations now in progress on Looney creek of Poor fork. The greater portion of this preliminary development was conducted by me while in the employ of E. V. d'Invilliers, geologist and mining engineer, of Philadelphia, to whom I am indebted for permission to use the data thus obtained. In this division about 350 coal-openings on 14 different seams were measured; many were sampled for chemical analysis, and most of them were located by stadia, to determine their exact positions and elevations, and to define the structure of the area. Complete geologic sections of outcropping strata were compiled at numerous points, the coal-seam intervals being determined from stadia-levels and the intervening strata being measured barometrically. The geologic features of this Eastern Division have been illuminated by recorded observations so numerous as to guarantee the general correctness of deductions and correlations made within the field.

II. General Description.

The Black Mountain coal-district, as here considered, is an area approximately 20 miles long by 8 broad, lying in Harlan county, Ky., and extending NE–SW. along the boundary between that State and Virginia. A glance at any map of the eastern United States will show that this district, with the adjoining territory to the north and northeast, forms the largest area east of the Mississippi river undeveloped by railroads and without navigable streams. Situated a short distance west of

the Atlantic-Gulf divide on the rugged Cumberland plateau, it is far removed from the great industrial centers and markets of the Atlantic sea-board and the Mississippi valley, and peculiarly isolated from them by lack of both rail- and river-transportation. To this geographic isolation are due its sparse population, its industrial stagnation, and its practically undeveloped wealth of coal.

For many years the nearest railroad to the Black Mountain district was the Cumberland Valley branch of the Louisville & Nashville R. R., which followed the southern foot of the Cumberland and Stone mountains in Virginia from Middlesboro to the town of Big Stone Gap, thence up the North fork of Powell river, 14 miles, to a connection with the Norfolk & Western R. R. at Norton. This line ran for a long distance within 10 miles of the coal-field, from entering which it was effectually barred, all the way, by unbroken mountain-ridges; and hence it added but little to the accessibility and value of the Kentucky field.

Similarly, in recent years, the railroad now known as the Virginia & Southwestern was built from Bristol and the southeast through Big Stone Gap to serve mines along the southern flank of Little Black mountain, but was never extended into Harlan county. This road passed close to the most important break in the Cumberland-Little Black Mountain barrier (Morris gap, just north of the mining town of Keokee); but, owing perhaps to the absence of proven areas of thick coal on upper Clover fork, no attempt was made to enter the field by means of this gap.

In 1909, a considerable coal-territory west of Harlan, and also on upper Poor and Clover forks, was controlled by important interests, able and anxious to develop their holdings; and in the latter part of that year construction was begun by the Louisville & Nashville R. R. on a branch about 50 miles long, extending from Wasioto up the Cumberland river to Harlan, thence along Poor fork to Benham, the mining-plant of the Wisconsin Steel Co., 2 miles up Looney creek. This branch is now operating throughout its entire length.

Harlan, the county-seat, is the only village in this district. It has at present a population of but a few hundred. However, with the stimulus of the railroad and its advantageous location

near the center of a large, virgin coal-field, it should become one of the more important towns of the region.

The area, as a whole, is sparsely settled. As the principal industry in the past has been farming, habitations are confined almost exclusively to the valleys, where the narrow bottoms and lower mountain-sides are under cultivation, leaving the long, steep upper slopes with much of their original forest covering.

From the forests the finer soft woods—poplar, lynn, ash, etc.—have been very generally cut and floated to the mills at Wasioto. But there still remains a vast amount of hard and semi-hard timber—the various oaks, sugar-maple, hickory, beech, and chestnut—only awaiting the advent of railroads to become valuable.

III. Topography.

Topographically, the district is one of great relief, with deep, contracted valleys separated by lofty ridges, the result of stream-erosion of an ancient, elevated, base-leveled plane. No plateaus exist; and, without exception, the level land at the mountain summits is confined to a strip from a few feet to a few hundred yards in width.

The declivities are usually very steep; the slope for hundreds of feet frequently exceeds 30°. At certain horizons—notably near the bottom of the Harlan formation—massive sandstone strata outcrop as great cliffs, forming conspicuous features in the landscape, as well as valuable guides to the position of certain coal-beds.

Topography has an important bearing on the economic value of this area; for the high ridges expose an unusual thickness of coal-measures and carry several valuable beds available for cheap drift-mining. The valleys form natural avenues for the location of railroads to serve the mines.

Erosion has, of course, greatly reduced the original areas of the higher coals. From the sharp-crested character of the mountains it follows that the remaining areas of the several beds vary inversely as their heights above the main drainage-channels.

Pine mountain is one of the chief topographic features of the region. Marking a great fault-line, its upturned strata of hard

sandstone and conglomerate rise northwardly from the valley of Poor fork at an angle of from 25° to 40° to a long straight crest more than 1,000 ft. above drainage. For 85 miles, from the Breaks of Sandy on the NE. to Pineville on the SW., no low gaps break this great mountain barrier. It is crossed by few roads; and it effectually separates the communities along Poor fork from those on Kentucky river to the north.

Big Black mountain, higher and even less passable than Pine, extends easterly from Harlan 40 miles to East Butte knob at the head of Poor fork, and forms the central divide, or "back bone," of the district. Throughout this distance it is from 3,000 to 4,000 ft. above tide, or 2,000 ft. above Poor and Clover forks. It is crossed by several bridle-paths but by only one wagon-road—the highway from Stonega, Va., to Whitesburg on the Kentucky river.

Thus, until the advent of the railroad, the central Poor Fork valley, in the vicinity of Looney and Clover Lick creeks, was peculiarly inaccessible, and all freight was wagoned from Stonega—a distance of 25 miles—over roads almost impassable, except in dry weather.

Benham spur and Looney ridge are the only noteworthy laterals on the Kentucky side of Big Black mountain. They are separated by Looney creek, and have the same general character and elevation as the parent mountain.

Little Black mountain, the southern boundary of the district, extends 25 miles in an almost straight line from Harlan to its junction with Big Black at "The Double," the highest knob of the region, more than 4,100 ft. above sea-level. At the west it divides the Clover and Martin forks of Cumberland river; on the east, Clover fork and the North fork of Powells river. High and rugged west of Childs creek, it sinks, along upper Clover fork, to a comparatively low ridge crossed in two places by wagon-roads. The more important of these passes is Morris gap, just north of Keokee, 2,622 ft. above sea-level, 520 ft. above Clover fork, and 590 ft. above the railroad grade at Keokee. Through this gap passes all the hauling between the railroad and the upper Clover fork country.

The following are the approximate elevations of some prominent points in the Eastern Division, determined by stadia-surveys, based upon an Interstate Railroad bench-mark on the

root of a maple 100 ft. north of the railroad station at Keokee, which bears the record of an altitude of 2,028.81 ft.—presumably above mean sea-level:

	Altitude. Feet.
Morris gap,	2,620
Clover fork, mouth of Left fork,	2,100
Clover, mouth of Huff creek,	1,945
Clover, mouth of Breeden creek,	1,680
Clover, mouth of Fugitt creek,	1,470
Summit Big Black, Fugitt-Pounding Mill trail,	3,330
Summit Big Black, Breeden-Trace trail,	[illegible],470
Summit Big Black, Huff creek trail,	3,615
Clover Lick creek, mouth Huff branch,	1,820
Clover Lick creek, mouth Peal branch,	,575
Clover Lick creek, mouth Pounding Mill,	1,460
Poor fork, mouth Clover Lick creek,	1,400
Poor fork, mouth Looney creek,	1,420
Looney creek, mouth Maggard branch,	1,535

IV. Drainage.

This feature of the district can best be understood by an examination of the map, Fig. 1. In the Western Division the drainage has been taken from the rather inaccurate "Jonesville" sheet of the U. S. Geological Survey; but in the Eastern Division the important streams were mapped from stadia-surveys, made in the course of the recent development-work.

The entire area is drained westwardly by the three head-tributaries of the Cumberland river—Poor, Clover, and Martin forks. To the north, beyond Pine mountain, the drainage flows in a northerly direction, and is carried to the Ohio by the widely-branching Kentucky river, while southward the Tennessee river receives the waters through its tributaries, the Powell and the Clinch.

Poor fork, heading against Pound river of the Big Sandy, follows an almost straight course along the foot of Pine mountain to the town of Harlan, where it joins Clover and Martin forks to form the main river. Though generally fordable, it has an abundant flow of water at all seasons, and meanders through bottoms averaging perhaps 0.25 mile in width, with a gradual fall of less than 1 per cent. Hence, a standard railroad could be constructed practically to its source; and many sites are afforded for mining-plants and saw-mills, to develop the great coal- and timber-resources of this section.

Looney and Clover Lick creeks, entering Poor fork 22 miles from Harlan, are its most important tributaries. Both are per-

ennial streams with gradual fall and narrow bottoms for several miles up from their mouths, giving ready access to the extensive areas of flat, outcropping coal in the heart of the field.

Clover fork, about 30 miles long, occupies the geographical center of the Black Mountain district. While similar to Poor fork in rate of fall and valley-contour, it drains a larger portion of the area under review. It is farther from the zones of upturned strata, has more large tributaries than Poor fork, and affords access to a very large coal-territory available for the cheapest form of self-draining drift-mining.

Martin fork skirts the field for a few miles SE. of Harlan, and is of interest chiefly as rendering a considerable area of the valuable Harlan coal-seam accessible to inexpensive mining.

V. General Geology.

The Black Mountain district occupies the eastern end of the Cumberland Gap coal-field, an area of coal-bearing formations 90 miles long by 10 to 20 miles wide, on the eastern margin of the great Appalachian field. Immediately south lies the wide area of older, non-coal-bearing rocks forming the Appalachian valley, while north and northwest the almost level strata of the coal-measures stretch away for many miles towards the limestone "Blue Grass" region of central Kentucky.

The outcropping rocks of this coal-basin are all sedimentaries. On the evidence of their fossils, they have been assigned by the U. S. Geological Survey to the Pottsville Formation of the Pennsylvanian Series, at the base of the Coal Measures.

It is believed that the lowest strata here exposed—the Lee Conglomerate, forming the crest and southern slope of Pine mountain—are at the bottom of the Pottsville, and the highest rocks on Black mountain are near the top of that formation, giving it a total thickness in this field of over 4,000 ft. As most of the strata lie almost flat, and are well exposed on the steep mountain-sides, the district offers unusual opportunities for detailed study of the Pottsville in this, its greatly thickened southern extension.

The SW. continuation of this district—from Harlan to Middlesboro—has been mapped and studied geologically by the

National and State governments[1] within recent years, and at a much earlier date (1887) by Messrs. E. V. d'Invilliers and A. S. McCreath.[2]

From Messrs. Ashley and Glenn's clear descriptions and numerous illustrations it has been easy to correlate the principal strata of the western field with those of the Black Mountain district, especially as the two areas overlap for a few miles just east of Harlan. The sub-divisions of the formation made by these observers have not been carried through the present description, but the limits of each are generally well defined, and are indicated on the suite of vertical geologic sections, Fig. 2. These sections were carefully compiled at points indicated by their headings and by letters corresponding with those on the accompanying regional map. In the case of those personally measured, intervals between coal-beds were deduced from stadia-locations and elevations, except the Jones creek section, where they were determined barometrically.

A glance at the geologic sections shows the strata to be sandstones and shales—with all intermediate grades of sandy shales and shaly sandstone; numerous coal-seams; and one thin, but geologically interesting, bed of limestone.

Sandstones largely predominate, especially in the upper part of the column, composing perhaps two-thirds of the total section above the Harlan, or *A*, coal. Though rarely conglomeratic, these beds are frequently coarse and massive, and form prominent cliffs on the mountain-sides, especially when underlain by softer, shaly strata, which disintegrate more readily and permit the massive beds above to break in great vertical faces.

The most noteworthy sandstone formation in the district begins at the top of the heavy shale-bed carrying the Fossil Limestone and extends upward about 800 ft. to the mountain-summits. This about corresponds to the Harlan Sandstone of Campbell.[3]

The lower 300 ft., in which occur the three High Splint coals,

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[1] G. H. Ashley and L. C. Glenn, *Professional Paper No.* 49, *U. S. Geological Survey* (1906).

[2] *Resources of the Upper Cumberland Valley:* published report to Louisville & Nashville railroad (1887).

[3] M. R. Campbell, *Bulletin No.* 111, *U. S. Geological Survey* (1893).

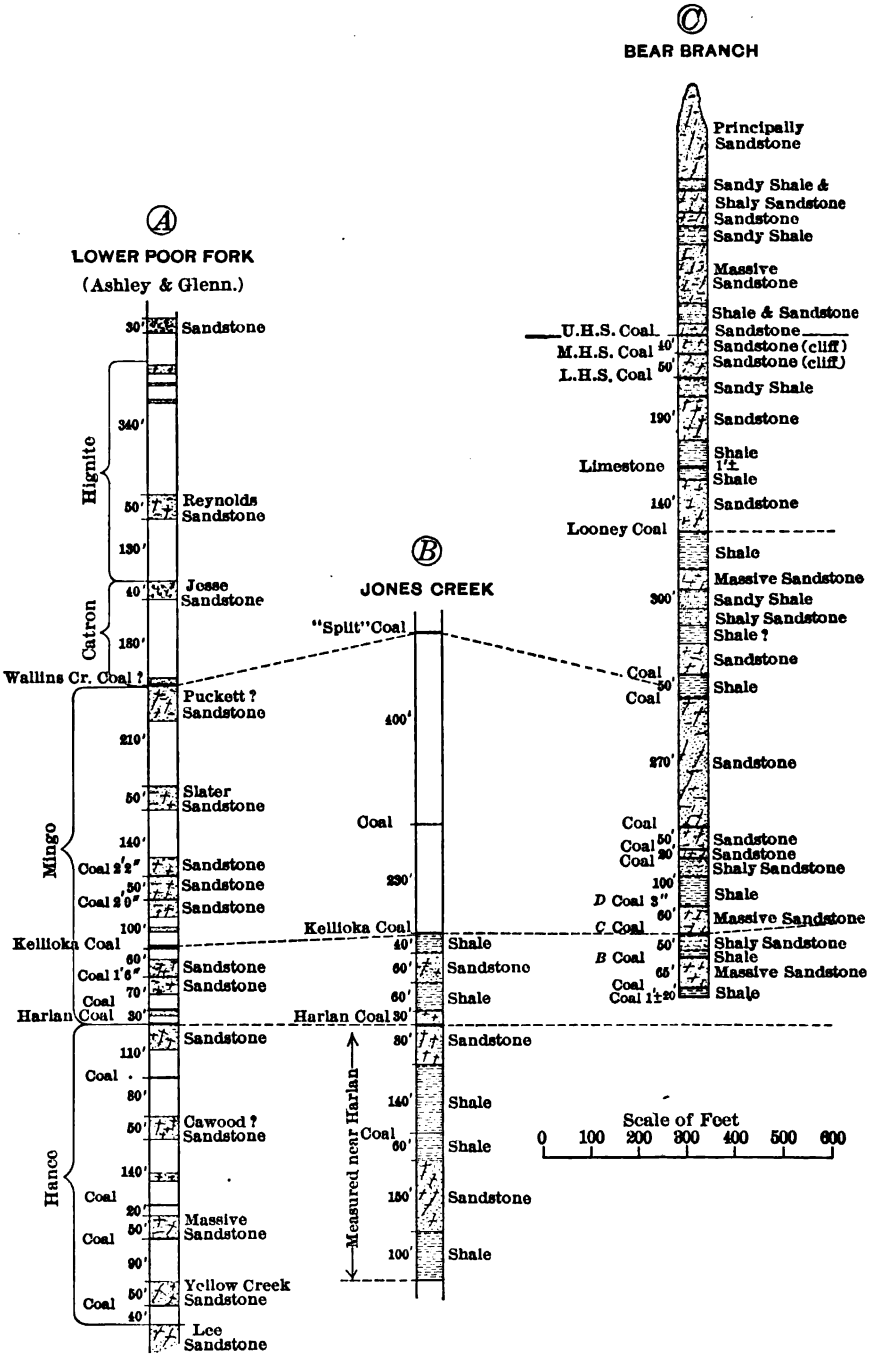
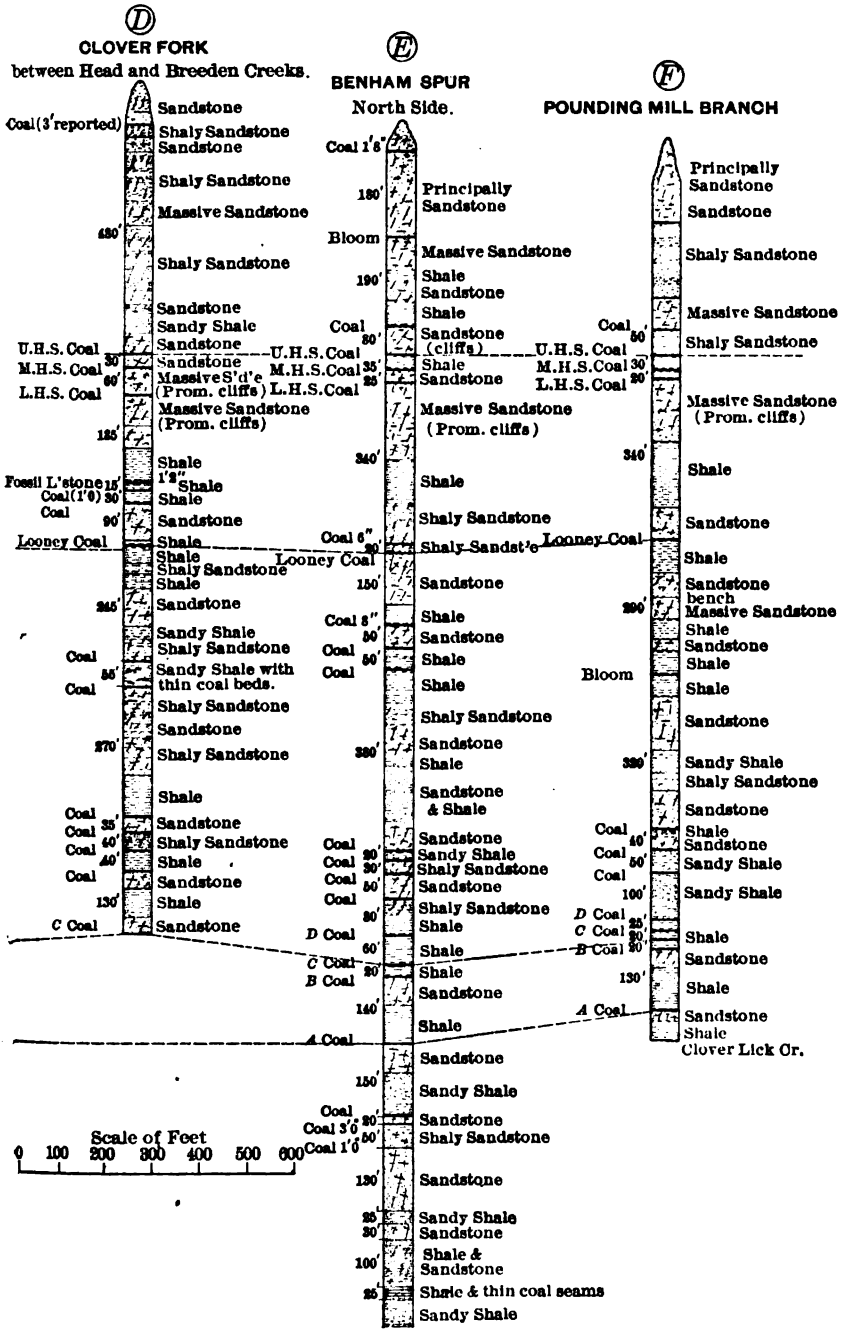


FIG. 2.—COMPARATIVE GEOLOGIC SECTIONS,



BLACK MOUNTAIN COAL-DISTRICT.

is particularly massive and forms the bold, gray cliffs so conspicuous on upper Clover and Poor forks along the brows of Big Black mountain and Benham spur.

Another sandstone formation of geologic interest lies between the Harlan (*A*) and the *B* coal-beds. It has been correlated by Campbell with the Gladeville, a widely recognized key-rock in the Virginia area to the south and east; but in the light of recent developments it seems probable that this formation is much nearer the Lee Conglomerate than the Gladeville Sandstone of his section. It is approximately 130 ft. thick, massive sandstone at top and shale or shaly sandstone at middle and bottom. The prominent outcrop of the sandstone member may be traced from Harlan, where it is more than 500 ft. above drainage, up Clover fork to Wynn branch. There it sinks beneath drainage, but, rising eastwardly about with the stream, it reappears below Head creek and forms the valley-bottom, to and above the Left fork. Its position in the upturned strata along the south bank of Poor fork is not so easily determined; but it is well developed on the Clover Lick and Looney tributaries of that stream.

Persistent shale-beds are rare in these measures. The most prominent noted is in the eastern part of the field, where it occupies an interval of about 100 ft. between the Harlan Sandstone above and a rather persistent sandstone—possibly the Reynolds of Ashley and Glenn—below. The rock is a soft yellow and gray shale, much less arenaceous than the prevailing rocks of the district.

The only known limestone stratum outcropping in this region occurs near the bottom of the above-mentioned shale formation, where it was seen at several places on upper Clover fork. A dark, impure bed about 1 ft. thick, it is remarkable for its abundance of fossil shells; but unfortunately only a small sample of the rock was obtained, and no careful study of its fossils has yet been made. This unique bed, if persistent, should give great assistance in correlating the strata of this district with those of Wise and Dickenson counties, Va.; for though it is thin, and occurs high in the hills, its peculiarities render it conspicuous, and many of its outcrops are known to the keen-sighted mountaineers.

Correlation in this region is difficult, by reason of the ab-

sence of any prominent and persistent key-rock. The massive sandstones frequently become shaly and insignificant or entirely disappear in short distances; the shale-beds are few and variable; and even the coal-seams—perhaps the best guides—can only be exposed by great labor and expense, and then vary between wide limits in section and interval. The great variability of the lower strata outcropping near the general drainage-level is well illustrated in the geologic sections, Fig. 2, where little harmony can be distinguished in the strata below the Looney coal-seam.

The coal-beds, to which this region largely owes its economical value, will be discussed in a separate chapter, after a brief consideration of the geologic structure of the field.

VI. Structure.

The Cumberland Gap coal-field is structurally a great trough or syncline, lying between the Pine mountain fault and the Powell valley anticline. The axis of this Middlesboro syncline, as it has been called, or the central line towards which the measures on either side dip, lies near the northern or Pine mountain side of the field, and closely follows Big Black mountain from Harlan to the head of Fugitt creek.

In the Western Division the strata on both banks of Poor fork, along the northern margin of the syncline, are sharply upturned, but flatten rapidly to the south as they approach the center of Big Black mountain. Along Clover fork, which drains the heart of the basin, the beds lie almost flat, having a northward inclination so slight as to require an instrumental survey for its detection.

Only in the Eastern Division were the locations of outcrops sufficiently accurate to permit a detailed exposition of the structure. The axis was found to follow a sinuous course, coincident with upper Fugitt creek; thence to extend eastward, crossing Clover Lick creek at the mouth of Cave branch, and cutting diagonally across Benham spur to intersect Looney creek about 5 miles from the river. In this distance the axis shows a decided pitch to the west, falling about 0.5 per cent., or 200 ft. in the 8 miles.

The measures along Poor fork dip southeast at angles of from

20° to 30°. The steep dips extend about 0.5 mile south from the river; then the beds flatten quickly to an almost horizontal position.

This 0.5-mile zone of upturned strata on the south side of Poor fork seems to follow closely that stream and to be independent of the position of the axis. On upper Fugitt creek, it is within 0.75 mile of the axis, towards which the strata dip somewhat rapidly, while on Looney creek the two have swung more than 2 miles apart and are separated by a wide area in which the average dip is about 2 per cent.

South of the axis the dips are somewhat irregular in direction and intensity, but never sharp. A subordinate trough appears to branch from the major one about central Fugitt creek, and to pursue a SE. course to the mouth of Breeden creek, with gentle dips, averaging less than 2 per cent. The beds lie almost flat in the Big Black mountain between Fugitt creek and Trace branch. The valley of middle Clover Lick occupies another minor trough of the main basin. From Breeden creek eastward to Huff creek the rise is irregular in direction and much more rapid, averaging about 3 per cent.

The structure of a coal-field is an important economic feature, since only with a knowledge of the direction and magnitude of bed-inclinations and the position of the major lines of elevation and depression can mines be properly laid out. Other conditions permitting, a mine should enter the coal at the lowest point of the area to be worked, that grades may be in favor of the loaded cars, and the drainage may be handled by gravity.

Several such favorable points of attack present themselves in this field, where gentle dips are the rule, and the principal beds are above drainage and suitable for cheap drift-mining.

VII. Geology of the Coals.

The geologic sections previously mentioned show the large number of coal-seams exposed in the 2,000 ft. of strata outcropping between the base and summit of Big and Little Black mountains, and indicate the relative stratigraphic position of each. They aggregate almost a score, of which about half have commercial value in individual parts of the field. Commencing at the lowest, these coal-beds will be dis-

cussed in ascending order, and from west to east, giving them the names by which they are known in their field of best development.

In the Lee Conglomerate, coal-seams are known to occur, but they are sharply upturned and economically worthless, and therefore will receive no consideration here.

In the several hundred feet of strata between the Lee and bed *A*—exposed along Poor fork and around Harlan—no coals of importance are known. On lower Martin fork, a bed from 2 to 3 ft. thick, about 220 ft. beneath the Harlan, or *A*, seam, has been opened in a few places; and along the south side of Poor fork, below Clover Lick creek, a coal apparently 50 ft. higher in the measures shows a generally thin and worthless section.

The absence of thick coals in this interval is of interest mainly as indicating that the famous Imboden seam of Wise county, Va., which occurs in this horizon, has deteriorated northward and either vanished or become insignificant in the Poor fork area.

The Harlan seam—or bed *A*, as it is named on upper Poor fork—is the lowest commercially-workable bed of the district, and the most valuable. It will furnish for many years practically all the tonnage won from the western ends of Big and Little Black mountains; and in the eastern end of the field it shows a good mining-section for a number of miles along Poor fork and tributary creeks. This seam, the outcrop of which is indicated by a dash line on the map, Fig. 1, has been opened extensively on lower Poor, Clover, and Martin forks, where it crops from 400 to 500 ft. above drainage, yet has more than 1,000 ft. of cover and underlies a large area in the Black mountains. West of a meridian line through the mouth of Jones creek, it shows a clean, uniform 4-ft. mining-section, below which is sometimes found another 18 in. of coal separated from the upper by several inches of shale. East of this line, along Clover fork, the bed is seriously contaminated by shale partings, and has been sparingly opened until it approaches water-level, 14 miles from Harlan. Here it presents a more attractive section, showing 3 ft. 6 in. of clean coal where it goes under the river just above the mouth of Seagrave branch.

Above this point the Harlan (*A*) seam does not outcrop on

Clover fork, but, rising eastward almost as rapidly as that stream, its horizon is never far below water-level—perhaps 20 ft. at the mouth of Fugitt creek, and 120 ft. at the mouth of Left fork, near Morris gap.

At Keokee, on the south side of Little Black mountain, it shows 7 ft. of coal with 15 in. of shale about 1 ft. from the floor, and is mined as the "Wilson" seam.

Along the main valley of Poor fork throughout the Eastern Division, bed *A* outcrops well above the stream, usually near the southern edge of the zone of upturned strata. In this area it exhibits a great thickness (12 ft. in some places), but the upper portion is usually slaty. However, below Looney creek the mining-section averages from 4 to 5 ft. thick, and will eventually yield a large tonnage. Above that stream the bed is apparently split into two benches, neither of which is especially valuable.

Bed *B*, lying 140 ft., more or less, above the Harlan, has no importance in the Western Division. It has been doubtfully recognized at an opening 3 miles up Yocum creek on the north side, showing 3 ft. 4 in. of coal interspersed with 1 ft. of shale, and is again seen with similar sections at openings on either side of Clover fork, 1.25 miles above Seagrave branch.

In the Eastern Division this is the most persistent though not the most valuable of the lower seams. Along Clover fork it is generally too thin or carries too much intercalated shale to be commercially important. Lying 120 ft. higher than the river at Fugitt creek, and very impure, it is opened where it goes beneath drainage 0.5 mile above Wynn branch, showing 3 ft. 6 in. of coal with two 5-in. partings in the upper half. But 4 miles further up stream it appears with a 2 ft. 2 in. section, and is frequently opened close to water-level from there to the mouth of Left fork, usually showing a thick section of alternate shale and coal benches, the latter generally too thin to mine.

There is an exceptional exposure of bed *B* in a country-drift, just above Head creek, where it measures 9 ft. 6 in.; the mining-bench, 4 ft. 3 in. thick, is at the bottom, with a 6-in. shale parting 9 in. from the top.

In Looney ridge and Benham spur this seam attains its best proportions, averaging about 3 ft. 6 in. of practically clean coal.

Its outcrop extends more than 4 miles up Looney creek and 3 miles up Clover Lick, offering large areas, very accessible to drift-mining. Numerous openings along the foot of Big Black mountain for several miles below Clover Lick creek indicate no value for the bed in that section.

Bed *C*—the Kellioka of Ashley and Glenn [4]—occurs approximately 30 ft. above the *B* seam. Geologically the most interesting seam of the field, because of its great areal extent and local peculiarities of bed-section, it is also, in certain localities, one of great commercial value.

Around Harlan this coal lies high in the mountains and has been but sparingly opened, as the more accessible and probably thicker Harlan, or *A*, seam supplies most of the local fuel.

Passing eastward to Jones creek, where the *A* seam begins to deteriorate, bed *C* has been opened frequently; and on Jones, lower Yocum, and Bailey creeks it shows a mining-section from 3 ft. 6 in. to 4 ft. thick, occasionally overlain by shale carrying thin layers of coal. In this area, embracing several thousand acres in Big and Little Black mountains, the *C* coal is better than anywhere else in the Western Division, and redeems a section practically devoid of other commercial beds.

Ascending Clover fork, its horizon gradually approaches the stream; sinks beneath it 0.75 mile below Breeden creek; reappears for a few hundred yards at the mouth of that creek; again emerges 1.5 miles further east, and stays close above drainage from there to and beyond the Left fork.

Scattered openings in this bed along Clover fork from Seagrave to Huff creek generally show a mining-section less than 3 ft. thick, sometimes overlain by a bench of shale and shaly coal, and occasionally by cannel or cannel slate—as at openings 0.5 mile up Fugitt creek, and on the south side of the river below Joe branch.

Above Huff creek, bed *C* shows a thickening section and loses its shale partings, until openings near Morris gap and opposite Keokee show over 5 ft. of practically clean, good coal. This condition is apparently maintained over large areas to the south and east; for the *C* seam is certainly the principal bed mined at Keokee (there called the McConnell) and most prob-

[4] *Professional Paper No.* 49, *U. S. Geological Survey* (1906).

ably the Taggart, worked at Roda in Wise county, Va. At both of these mines it is from 5 ft. to 6 ft. 6 in. thick. In the Pocket district of Virginia it is probably this seam which is mined as the Darby, and shows from 2 ft. 6 in. to 3 ft. 6 in. of clean coal.

In Looney ridge and on the east side of Benham spur, bed *C* attains its greatest development in the Black Mountain district. Throughout this area it is uniformly a seam of the highest commercial value, averaging 5 ft. 6 in. in thickness, and practically without partings.

In Benham spur a remarkable and geologically interesting feature is presented in connection with this bed. In tracing its outcrop around the nose of the spur, a small shale parting near the center of the section is first observed at an opening near the mouth of Looney creek; in a few hundred yards this parting has swelled to 1 ft.; and when Clover Lick creek is reached the two coal-benches are separated by from 20 to 50 ft. of sandstone and shale, the former very massive and making prominent cliffs along the creek. This great parting has but slightly altered the usual 80-ft. interval between beds *B* below and *D* above; consequently the lower bench of *C* is found within a few feet of *B*, and the upper bench close under *D*.

This split condition of bed *C* obtains on both sides of Clover Lick creek, and probably extends down Poor fork several miles. It is believed that the two benches unite under Big Black mountain, to form the comparatively thin seam identified as *C* on Clover fork.

Bed *D* is an insignificant and rarely-observed seam, except along Looney and Clover Lick creeks of Poor fork. No openings at its horizon were noted in the Western Division, and on upper Clover fork it shows only a few inches of coal.

Along lower Looney creek it lies 50 ft., more or less, over *C;* but on Clover Lick, where *C* is split by a great parting, the interval between *D* and the upper bench of *C* varies from 10 to 35 ft. It averages about 3 ft. 6 in. of clean coal at the many openings made, indicating large commercial areas in Looney ridge and Benham spur. Throughout this part of the field the seam is characterized by a stratum of slate and coal one or more feet thick, lying just under the main mining-section.

The next coals to be noted in ascending the geologic column

are a group of four beds occurring within a vertical interval of 120 ft., the lowest of which is about 130 ft. above the *C*, or Kellioka, seam. All are too thin to deserve more than passing notice.

In the Western Division a few scattered openings have been made in one or more of these beds; but it is impossible to correlate them definitely by reason of their similar thin sections and the small interval between them. To this group belongs the Creech (?) coal of Ashley and Glenn.[5]

Extensive prospecting in the Eastern Division has disclosed one or more of these four seams in numerous places; all four may be seen close to drainage on Breeden creek, near the mouth of Right fork.

The bottom bed is very persistent, and wherever opened shows about 2 ft. of clean coal. The second is some 40 ft. higher, and while showing a 3 ft. 4 in. mining-section on Looney ridge, it is generally of no value. The third is about 40 ft. still higher, and is worthless. The fourth, at the top of the group, has been recognized principally along Clover fork above Breeden creek, where it is mined at a few country pits. This seam, or the one next below, represents the so-called "Low Splint" bed, of regional extent and commercial thickness on the south side of Little Black mountain in Virginia.

Mention may be made of an opening three-quarters of a mile up High Bank branch, a small tributary entering Poor fork 6 miles below Clover Lick creek. The seam here exposed lies about 300 ft. above *C* and is locally known as the "11-foot cannel bed." In reality it consists of 7 ft. of cannel-slate or highly-bituminous shale, worthless for fuel, parted 2 ft. 10 in. from the roof by 4 ft. 4 in. of black shale, and underlain by another foot of that material. At no other place was this seam observed; it is evidently a locally bituminized shale-bed.

The Wallins creek coal, which attains such fine proportions on the stream of that name below Harlan, is represented in the Western Division of this district by a thick seam, sparingly opened high on the flanks of the Black mountains. It lies about 600 ft. above the Kellioka, or *C*, seam.

In the head of Kitts branch, entering Clover fork 2 miles

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[5] *Professional Paper No.* 49, *U. S. Geological Survey* (1906).

above Harlan, this bed shows an upper bench of 4 ft. 4 in. of coal with a 6-in. shale parting 8 in. from the floor, and a lower 1 ft. 2 in. coal-bench separated from the upper by 1 ft. 4 in. of shale, coal, and bone. On the left prong of Turtle branch it measures 4 ft. 8 in. of clean coal.

In the Eastern Division, near the Wallins creek horizon is usually found a mass of shales carrying two or more thin coal-beds (see geologic sections, Fig. 2). On upper Childs creek, the two seams are 15 ft. apart, the upper being 4 ft. 4 in. and the lower 3 ft. 11 in. thick, and both somewhat impure. They are again exposed near the mouth of Breeden creek, where one opening shows 3 ft. 9 in. of coal with 5 in. of shale; but elsewhere this horizon has afforded no coals approaching commercial character.

The remaining coal-beds to be considered lie high in the mountains; their outcrops are rather inaccessible and have been rarely opened, to supply local fuel-requirements. Consequently there is no knowledge of them in the Western Division, where they underlie relatively small areas, and can have but little economic significance.

The Looney bed, the lowest of these, is named from Looney ridge, its area of best development in the Black Mountain district. This seam occurs approximately 850 ft. above bed *C* and 135 ft. below the Fossil Limestone (page 140), and is probably equivalent to the Parsons seam of Wise county, Virginia.

In the high knob of Little Black mountain, at the heads of Yocum, Childs, and Straight creeks, the Looney seam has been opened in several places and shows from 4 ft. to 5 ft. 6 in. of coal, usually parted by a little shale. In Big Black mountain it is generally thin and worthless, though a limited area between Bear and Kelly branches will average 3 ft. of clean coal.

On the south side of Benham spur, it shows only a few inches of coal, but thickens northward until, in Looney ridge, a clean 5-ft. section is displayed.

The coals of the High Splint group are the highest and last beds to be mentioned; for though a few "blooms" or outcrops of beds have been noted in the 500 ft. of strata between this group and the mountain-tops, nothing is known of their character; and they are too inaccessible and too limited in area to have present commercial value.

The Lower High Splint bed lies from 300 to 350 ft. above the Looney seam, separated from the Middle by from 30 to 60 ft. of rock—along Clover fork usually a bold cliff-making sandstone. It varies greatly in bed-section. Valueless in Looney ridge and Benham spur, this seam shows in Big Black mountain more than 3 ft. of minable coal, frequently overlain by from 2 to 3 ft. of slate and coal in alternate thin benches.

The Middle High Splint bed, 30 ft., more or less, beneath the Upper, is a quite clean and persistent, though uniformly thin seam, averaging about 3 ft. in Big Black mountain and somewhat less in Benham spur.

The Upper High Splint bed occurs uniformly 400 ft. above the Looney, and is by far the most valuable of the high seams in the Black mountains. It is known to exist with splendid section over large areas east of this district, and, though rather inaccessible to mining-operations, will undoubtedly supply a large tonnage of superior gas- and domestic fuel. Its approximate outcrop is delineated on the map, Fig. 1.

The seam lies too high above drainage to have any appreciable area in Little Black mountain. Big Black mountain, by reason of its superior height and its position near the axis of the coal-basin, contains the largest and most important area of Upper High Splint in the district. It outcrops from 1,000 to 1,300 ft. vertically above the valleys of Clover and Poor forks.

In the eastern portion it averages about 6 ft. of coal, devoid of serious partings. Westwardly its section diminishes somewhat, and numerous measurements on Fugitt and neighboring creeks average slightly less than 4 ft. of clean coal. That the seam again thickens below this area is indicated by an opening on Joe branch, showing 9 ft. 5 in. of coal with a 6-in. shale parting 1 ft. from the floor. However, it is near the mountain-tops here, and probably underlies no important areas west of this point.

Benham spur carries a good acreage of this bed averaging over 4 ft. in thickness. Its condition in Looney ridge is unknown, as the portion of that mountain high enough to carry appreciable areas of the High Splint beds lies east of the field under review.

VIII. Character of the Coals.

Chemically, the coals of the Black Mountain district are very pure, high-volatile gas- and coking-fuels, as evidenced by the proximate analyses, given in Table I. The samples affording these analyses were taken from outcrop openings by cutting a uniform amount of material from top to bottom of the bed, excluding only such slate partings or coal-benches as should be discarded in mining. They were very generally obtained from freshly-prepared faces under solid roof, where the coal was hard, clean, and lustrous. The resultant analyses should fairly represent the regional quality of the seams; and, except for excessive moisture in some instances and a slightly-diminished sulphur-content, due to leaching, they should be quite comparable with those made from mine-samples.

Table I.—*Coal Analyses: Black Mountain District, Harlan County, Ky.*

Bed.	Division.	Water.	Volatile Matter.	Fixed Carbon.	Ash.	Sulphur.	Phosphorus.
A, or Harlan[a]	Western	1.48	38.53	57.56	3.91	0.87	0.0039
A, or Harlan[b]	Eastern	1.23	39.19	54.72	6.09	1.27	0 0037
B [c]	Eastern	3.32	38.72	57.95	3.33	0.65	0.0057
C, or Kelliokа[d]	Western	1.57	38.41	58.89	2.70	0.64	0.0032
C, or Kelliokа[e]	Eastern	3.42	38.79	58.92	2.29	0.65	0 0036
D[f]	Eastern	1.51	36.06	53.73	10.21	0.79	0 0195
Looney[g]	Eastern	4.28	38.67	53.40	7.93	0.95	0 0080
Lower High Splint[h]	Eastern	4.82	38.79	57.19	4.02	0.76	0.0090
Upper High Splint[i]	Eastern	4.92	38.85	56.82	4.33	0.65	0.0200

a. Average analysis 21 samples.
b. Average analysis 6 samples.
c. Average analysis 16 samples.
d. Sample from Yocum creek.
e. Average analysis 17 samples.
f. Average analysis 13 samples.
g. Average analysis 3 samples.
h. Average analysis 7 samples.
i. Average analysis 11 samples.

A striking feature of Table I. is the marked similarity in analyses of coals from widely-separated geologic horizons. It will be noted that the *B* and Upper High Splint beds, 1,300 ft. vertically apart, yield fuels of almost identical chemical composition.

The comparatively high ash found in the *D* and Looney beds is perhaps due rather to small slate binders included in the

sample than to any inherent difference in the coal itself. The generally low percentage of ash and sulphur in the principal seams is noteworthy and adds greatly to their commercial value. Ash is remarkably low in the *C*, or Kellioka; its percentage in 18 samples falls between 1.45 and 4.25, classing this bed with the purest coals known.

Considering the seams as a whole, they are typical gas-coals in chemical composition; high in volatile hydrocarbons essential for the manufacture of illuminating-gas; low in the harmful ingredient, sulphur, and exceptionally free from earthy impurities.

As domestic and steam-fuels they should rank high. Their hardness, ease of ignition, low ash, low sulphur, and high calorific power are all features which will commend themselves to consumers. Calorimeter-tests of carefully-selected samples from the more important beds showed about 14,000 B.t.u. per pound of coal.

Coking-tests of crushed coal from the lower seams have given very satisfactory results, as was to be expected, since the Roda and Keokee plants, operating the *C* seam in Virginia, manufacture a highly-valued metallurgical coke.

Physically, the coals of this district are hard, lumpy, and of the block type, similar to those of the Kentucky River and Elkhorn fields. Characterized by marked vertical cleavages, they mine in large, smooth-faced blocks, very different from the softer, columnar coals of the Pocahontas and New River districts in West Virginia, and better adapted to shipment and domestic use. In the Eastern Division of the district, the major cleavage-planes, or "faces," have a uniform direction of about N. 65° E.–S. 65° W.

Two varieties of coal, quite different physically, are found in varying proportions at almost every exposure in the field. One is a hard, dull gray splint that occurs in layers varying in thickness from a fraction of an inch to a foot or more. The thicker layers are exceedingly tough and hard, mining in large blocks which themselves break with a conchoidal fracture into sharp-edged fragments.

The other is a softer, black, lustrous variety, which breaks readily into small cubical pieces. It mines more easily than

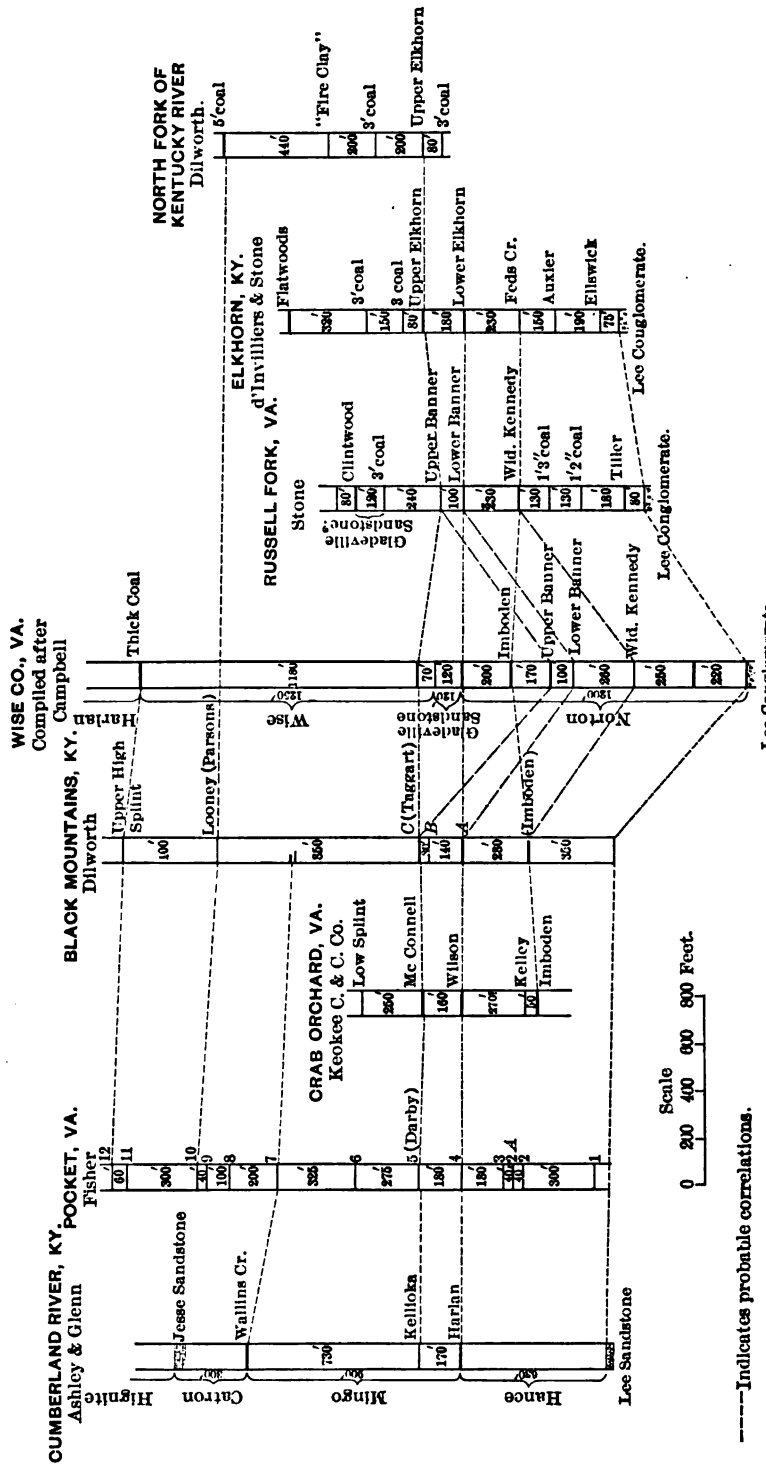

FIG. 3.—COMPARATIVE GEOLOGIC SECTIONS, SUGGESTING CORRELATIONS.

the splint; is less difficult to crush, and is more typically a coking-coal.

No one bed is all splint or all coal of the softer variety; neither do any two openings, even in the same seam, show the same proportion of these constituents. But, as a rule, more splint is found in coals of the High Splint group, the gas- and domestic fuels, than in the lower beds, *A–D*, which are more truly the steam- and coking-seams.

IX. General Correlations.

After the foregoing details of the several coal-beds of the district, some brief suggestions of their relations to other well-known seams in neighboring fields are pertinent. General vertical sections of the measures exposed in these fields are shown graphically in Fig. 3.

As already remarked, there is no difficulty in correlating the coals of the Black Mountain district with those of the territory to the west, where the important *A* and *C* beds are represented by the persistent Harlan and Kellioka seams.

To the south, in the Pocket district, Virginia, the identification of the Darby, or No. 5 seam of Fisher,[6] as bed *C*, is substantiated by a comparison of general geologic sections on either side of Little Black mountain; by the relative elevations of outcrop openings; and by the character of the seam in each locality. The equivalency of No. 10 and Looney, and of No. 12 and Upper High Splint, is similarly indicated.

In the Crab Orchard district, just east of the Pocket, the two principal coals, Wilson and McConnell, are, in all probability, the representatives of beds *A* and *C*.

The territory to the east, in Wise county, Va., was investigated by E. V. d'Invilliers and A. S. McCreath before the advent of railroads; and again by M. R. Campbell,[7] of the U. S. Geological Survey, a couple of decades ago, when it was sparingly developed, and the poorly-exposed geologic horizons were traceable only with great difficulty. The northwestern portion of Mr. Campbell's field of investigation overlaps the present one, but it is difficult to harmonize the results obtained in each. From

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[6] Cassius A. Fisher, *Bulletin No.* 341, *U. S. Geological Survey* (1907).

[7] Marius R. Campbell, *Bulletin No.* 111, *U. S. Geological Survey* (1893).

his descriptions it is evident that coals he finds just above and below the "Gladeville Sandstone" on (Big) Looney and Clover Lick creek are beds *A* and *B*. It is also evident that his thick coal at top of the Wise formation is the Upper High Splint.

Developments of the Virginia Coal & Iron Co. in Wise county, Va., whereby the outcrops of the High Splint (Upper, probably), Parsons (Looney), Taggart, and Imboden seams have been traced over large areas, render almost certain the identity of *C* and Taggart, or Roda, coals, and place the Imboden seam 450 ft. below this horizon, or 280 ft. under bed *A*, about where Campbell locates it. Thus far there is seen to be a close agreement between Campbell's general section and the one given for the Black Mountain field.

The great discrepancy appears upon comparing the interval he found between the Imboden bed and the Lee Conglomerate with that deduced from Messrs. Ashley and Glenn's measurements[8] in the Black Mountain field. The former made it 1,000 ft.; the latter, 350 ft.

Considering now that portion of the geologic section just over the Lee Conglomerate, we find both Mr. Campbell and Mr. Stone[9]—the latter working in the NE. extension of the Wise County field on Russell fork waters—locating the Kennedy bed 500 ft., more or less, above the Lee, with the Lower and Upper Banners respectively 250 and 350 ft. still higher in the measures. The correlation of these beds with the Imboden, *A*, and *C* (or *B*) seams of Harlan county suggests itself.

To harmonize the correlations deduced as above, it is suggested that the coal Campbell called the Imboden (and clearly recognized only in the central portion of his field) might be the same as the Kennedy bed (recognized only in the eastern portion), and that an error was made in connecting the geology of these two divisions of his territory.

It would follow that the Gladeville Sandstone lies between the two Banner coals, or, if it be about 370 ft. above the Upper Banner, as shown in Campbell's section, the sandstone called by that name on Poor and Clover forks really lies that distance below the true Gladeville horizon.

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[8] G. H. Ashley and L. C. Glenn, *Professional Paper No.* 49, *U. S. Geological Survey* (1906).

[9] Ralph W. Stone, *Bulletin No.* 348, *U. S. Geological Survey* (1908).

In the Elkhorn, Ky., field, north of Pine mountain, correlations may be made as indicated on the plate of sections, Fig. 3, using the Lee Conglomerate, as located by Mr. Stone, as a base. The coal-bed intervals shown in the Elkhorn general section apply particularly to the Marrowbone district, a part of the field prospected in great detail some years ago by E. V. d'Invilliers, of Philadelphia, Pa.

The last general section represents the measures on the North fork of the Kentucky river, between its head and Hazard, the county-seat of Perry county, and was compiled barometrically several years ago by me. Strangely enough, it bears but slight resemblance to the geologic section of the Black Mountain district immediately south. Though the two areas adjoin, the various geologic horizons cannot be traced from one to the other, by reason of the great strata-displacement resulting from the intervening Pine Mountain fault. However, certain correlations are suggested on the plate of sections, carrying identification via the Russell Fork, Va., and Elkhorn, Ky., fields.

Reviewing the correlations here advanced for one of the principal coal-seams in the Black Mountain and adjoining fields, it seems most probable that the *A* bed of Looney and Clover Lick creeks of Poor fork is the Harlan of Ashley and Glenn; the No. 4 of the Pocket district, as given by Fisher; the Wilson of the Crab Orchard area; the coal recognized by Campbell on Poor fork as lying just under his "Gladeville Sandstone;" and that it is represented in the Tom's Creek field, Va., by the Lower Banner seam, and in the Elkhorn field, Ky., by the Lower Elkhorn.

Additional evidence of the approximate correctness of the identifications made is furnished by fossils associated with certain coal-horizons in the various fields.

It has been remarked by a competent palæobotanist that the base of the Wise formation (beds *A* to *C*) is perhaps equivalent to the Eagle coal-group of the Kanawha, W. Va., field, which group is believed to correlate with the Feds Creek seam of Kentucky, lying 230 ft. beneath the Lower Elkhorn and 400 ft., more or less, above the Lee Conglomerate. If this be even approximately true, bed *A* certainly lies no higher in the geologic column than the Lower Elkhorn, and could not be 1,200 ft. above the Lee, as shown in Campbell's section.

Again, David White, of the National Museum, says:[10]

"The present state of knowledge does not justify a correlation of the Elkhorn with the coal beds in the Norton or Big Stone Gap region. It may, however, be remarked that the species of fossil plants from the Lower Elkhorn appear to bear a close relation to such material as I have been able to secure from the Banner group, at Dorchester, near Norton."

This agrees with the correlation suggested for the Russell Fork and Elkhorn fields.

But it must be admitted that the suggested correlations between Black Mountain coals and those of the Norton, Russell Fork, and Elkhorn fields are supported by few unquestioned facts. And though all available data have been used to throw light on the subject, and the conclusions reached seem to be supported by a preponderance of evidence, there is need of much more detailed exploration of intermediate areas before the mutual relations of the important coal-horizons in the several fields can be positively determined.

In closing, it may be noted that, if the foregoing correlations are correct, the Pocahontas and New River coal-groups occur in the Lee Conglomerate, and are represented in this region by a few sporadic and worthless seams.

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[10] *Bulletin No.* 348, *U. S. Geological Survey*, p. 32 (1908).

The Geology of the Tonopah Mining-District.

BY AUGUSTUS LOCKE, GOLDFIELD, NEV.

(San Francisco Meeting, October, 1911.)

Two Opposed Interpretations of the Tonopah Structure.—The important geological publications concerning the Tonopah mining-district are those of Spurr[1] and of Burgess.[2] In these publications are presented fundamental differences of interpretation, which are the more interesting because both authorities have had ample opportunity for observation, and because both are geologists of proved ability. I was vastly puzzled to know which had the better of it, and in order that I might reach a conclusion in my own mind, I recently spent some time in the district going over the physical evidence. Surprising as it may seem, this evidence looks conclusive. A review of it will, I believe, be of interest to those who are familiar with the previous publications.

The general geological features of Tonopah are shown in Fig. 1, and the differences of interpretation referred to are outlined in the accompanying notes. Briefly, Burgess regards the various rocks as flows, lying in the order of their deposition. Spurr regards them in part as flows, and in part as flat-lying intrusives. The disagreement, then, concerns the rocks regarded on the one hand as intrusives, and, on the other hand, as flows. These rocks are chiefly the so-called calcitic andesite, the upper rhyolite, and the lower rhyolite.

Economic Importance of the Question of Interpretation.—The economic importance of the question of interpretation is, of course, limited to its bearing on the probable distribution of

[1] Geology of the Tonopah Mining District, Nevada, *Professional Paper No.* 42, *U. S. Geological Survey* (1905). *Report on the Geology of the Property of the Montana-Tonopah Mining Co.* (1910). An abstract of this report is given in the *Mining and Scientific Press*, vol. cii., No. 16, p. 560 (Apr. 22, 1911).

[2] The Geology of the Producing Part of the Tonopah Mining District, *Economic Geology*, vol. iv., No. 8, pp. 681 to 712 (Dec., 1909).

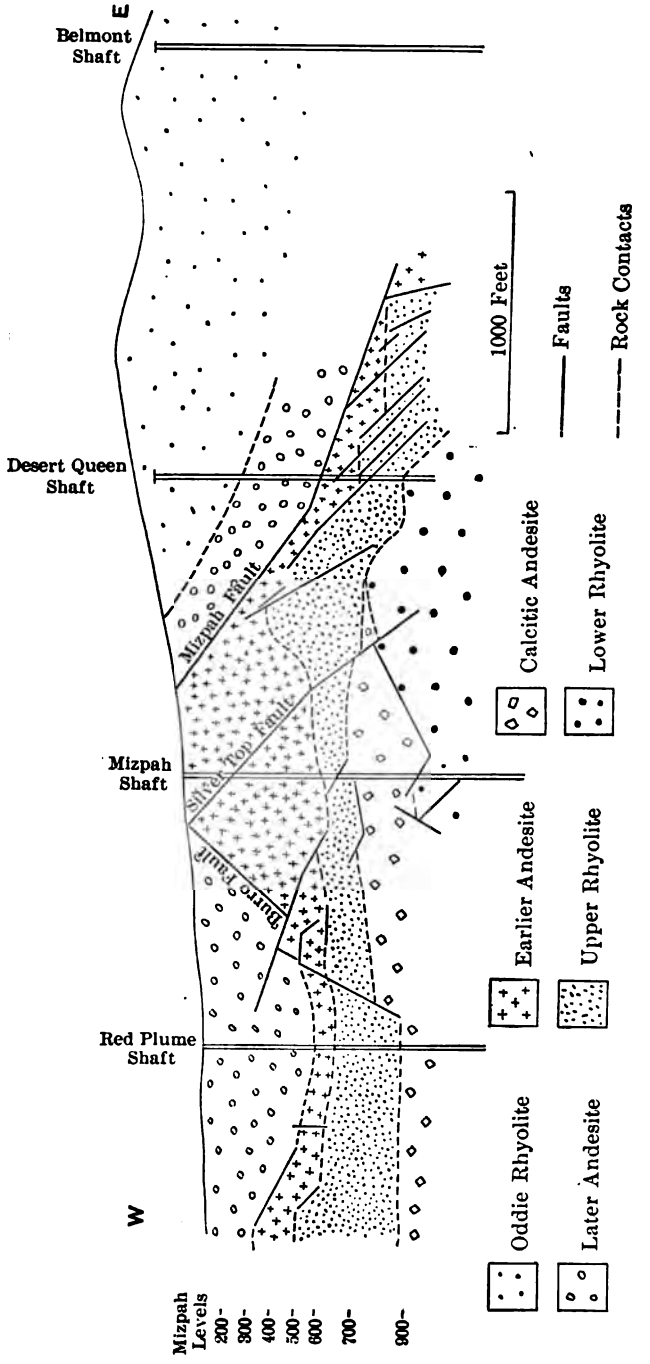

FIG. 1.—EAST-WEST SECTION THROUGH MIZPAH SHAFT.

Adapted from section by Burgess, *Economic Geology*, vol. iv., No. 8, p. 681 (Dec., 1909). Facts indicated by this section constitute one of the chief arguments in favor of the hypothesis that the rocks are flows, lying in the order of their deposition.

undiscovered ore. The later andesite is generally conceded to be barren—a "cap rock," at whose lower contact the productive veins apex. The earlier andesite has so far yielded the bulk of the production. As has been already suggested, both Spurr and Burgess regard it as a flow, and both have essentially the same conception of its distribution. Above the bottom of the earlier andesite, therefore, the conception of the ore-distribution is the same, whichever interpretation be adopted.

Below the bottom of the earlier andesite, however, the matter of interpretation assumes supreme economic importance; for, while Burgess regards all the underlying rocks as older than the earlier andesite, and older than the chief ore-mineralization, Spurr regards them as younger than both. Under Spurr's hypothesis, exploration in these rocks is emphatically discouraged; under Burgess's, it is to a certain extent encouraged.

Outline of the Evidence.—The important evidence appearing to favor the hypothesis that all the rocks occur in flows is as follows:

1. The locus of each rock is horizon-like. For example, the lower rhyolite is encountered at depths averaging about 1,000 ft., over an area of at least a square mile. Its surface, except where it is faulted, is seldom steeper than hill slopes, and is chiefly flat or horizontal.

2. Materials closely resembling stratified volcanic tuffs occur abundantly on the upper contact of the lower rhyolite, and less abundantly on the upper contact of the upper rhyolite.

3. The contacts between the supposedly intrusive and intruded rocks are, when unfaulted, most often notably straight and regular. Nowhere have the so-called intrusives been conclusively proved to invade by means of offshoots the rocks which they have supposedly intruded. The interpretation of irregularities of contact as proof of intrusion is made difficult by the abundant faulting, and by the possibility of inter-flow erosion.

4. The andesitic cover has, over a large area, rigidly confined the rocks which underlie it. The lower rhyolite, a rock having a very characteristic and unmistakable appearance, has been proved to occur on the surface only in the territory considerably north of the producing mines, and there in very small and scattered bodies which may be inclusions.

5. The productive veins in certain places pass without diminution either in size or richness from the earlier andesite down into the upper rhyolite.

6. In many places near the top and near the bottom of the upper rhyolite, there occurs an extraordinary igneous breccia, often many feet thick and crowded with foreign inclusions; the matrix is rhyolitic, and the rock looks exceedingly like a flow breccia. The upper portion of the lower rhyolite has numerous but less abundant inclusions.

7. The rhyolites, though containing abundant inclusions, and among them some which are andesitic, have never yielded inclusions which can be positively identified as belonging to the earlier or later andesites.

8. The andesites are free from inclusions of all sorts; therefore their freedom from inclusions of rhyolite is no indication that they are older than the rhyolites.

The evidence supposedly favoring the hypothesis that some of the rocks are intrusive is as follows:

1. In the rhyolites, a banding resembling flow structure sometimes follows irregularities in the contact.

2. The rhyolites occasionally have on their contacts with the andesites knob-like and wedge-like projections, looking like intrusive shapes.

3. In certain places, the calcitic andesite is separated by rhyolite from the later andesite with which it is supposed by Spurr to be identical. In certain places, the earlier andesite is separated by upper rhyolite from a rock called glassy trachyte, with which Spurr supposes it to be identical.

4. The profitable veins often disappear or weaken when they reach down to the lower contact of the earlier andesite.

The Evidence Afforded by the General Distribution of the Rocks.—It must be granted at the outset that the disposition of the rocks in horizons creates the presumption that they are flows. Most of the shafts penetrate similar rocks in similar succession. Thus, the lower rhyolite, so far as is known, underlies the whole district; the calcitic andesite almost everywhere covers the lower rhyolite; and above these rocks come, in order, the upper rhyolite, the earlier andesite, and the later andesite. The individual sheets of rock have many irregularities in

thickness; these, however, are satisfactorily attributable to inter-flow erosion and to faulting.

Again, if we conclude that the earlier andesite is the oldest rock in the district, we must conclude also that it has been floated up by the intrusive underlying rocks to a height of at least 1,000 ft., and possibly to a much greater height. (The lower contact of the lower rhyolite is not known.) During the process of floating up, the andesite has retained over an area of at least a square mile, its integrity and approximate horizontality.

A general view, then, of the large features of rock-distribution affords strong evidence in favor of the theory of extrusion. Nevertheless, it is conceivable that pseudo-flows might result from intrusion, and this evidence is therefore, by itself, inconclusive.

The Evidence Afforded by the Supposed Tuffs.—If the large facts of rock-distribution fail to furnish conclusive evidence of the origin of the rocks, this evidence must be sought in the details of the rock-contacts.

In general, there are certain details of rock-contacts the testimony of which must be accepted as unimpeachable. One such detail is the existence at contacts of volcanic tuffs; genuine tuffs being proved to exist between layers of volcanic rocks, it is difficult to conceive of evidence, however abundant, which would prove that the layers are not flows.

It becomes, therefore, a matter of extreme importance to determine whether or not in Tonopah the supposed tuffs are genuine. Burgess, who discovered them, believes that they are. Spurr believes that they are not.

That they are tuff-like, is beyond doubt. They are somewhat soft; they possess stratification, marked by alternating bands of coarse and fine fragmental material; they cleave easily along the junctions of these bands; they lie with their structure parallel with the rock-contacts.

If they are not true tuffs, resulting from surface deposition, then they are conceivably attributable to one or both of two processes—flow-banding (the arrangement of inclusions along flow-lines) and movement-banding. Spurr's conception of their origin is expressed in the following: ". . . brecciated and granulated rock is often layered by the fault-movement and

fault-pressure, so that it assumes all the appearance of certain varieties of surface-formed detrital tuffs."[3]

Microscopic examination of thin sections of specimens from the Mizpah 700-ft. level yields conclusive evidence against the possibility of the production of the supposed tuffs either by flow-banding or movement-banding. (1) The tuffs are made up of sharp-cornered fragments, often crowded closely together, and are typically clastic. (2) They are distinctly layered; layers of coarse material alternate with layers of fine material, with no gradation from coarse to fine. (3) The abundant quartz phenocrysts, with one or two exceptions, when revolved in polarized light, extinguish with much suddenness. The wavy extinction, which is the invariable characteristic of strained quartzes, is strikingly absent.

That a sorting-out and sharp separation of coarse from fine should result from flow- or movement-banding, is, of course, incredible. And the significance of the unstrained quartz phenocrysts cannot be questioned. Indeed, the tuffs are so life-like and their detrital origin so obvious that their import would be ordinarily accepted without question.

The Supposedly Intrusive Contacts.—The supposedly intrusive contacts of rhyolite with other rocks at no place seen by me offer incontestable evidence of intrusion. Before such proof can be accomplished, it is necessary to prove that the irregularities were not caused by faulting, or by inter-flow erosion, or by both. Now, in localities of extensive rock-alteration and abundant faulting, such proof is impossible; indeed, here the proof that the irregularities were actually caused by faulting is frequently possible.

The rhyolite at certain places possesses a banding which follows to some extent irregularities of the contact, and which sometimes looks like flow-banding, and might suggest intrusion. I have failed, however, to find any such place where the evidence of intrusion was unequivocal. Usually, the banding is irregular and very discontinuous. It is quite as often oblique to the contact as parallel with it. Moreover, if contact-movement, as Spurr believes, can produce tuffs, it is very easy to conclude that it can produce apparent flow-structures.

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[3] *Report on the Geology of the Property of the Montana-Tonopah Mining Co.* (1910).

Evidence Afforded by the Localization of the Profitable Ore-Deposits.—The usual localization of the profitable ore-deposits to the earlier andesite is one of the most interesting facts of ore-occurrence with which I am familiar. In certain cases, the ore ends abruptly when it comes down to the lower contact of the andesite. In other cases, it extends down into the underlying rhyolite, ultimately, however, weakening and dying out. Occasionally, see Fig. 2, it survives for a time with a hanging-wall of andesite and a foot-wall of rhyolite, ceasing shortly after it passes entirely into the rhyolite. Lastly, it passes from andesite to rhyolite without change.

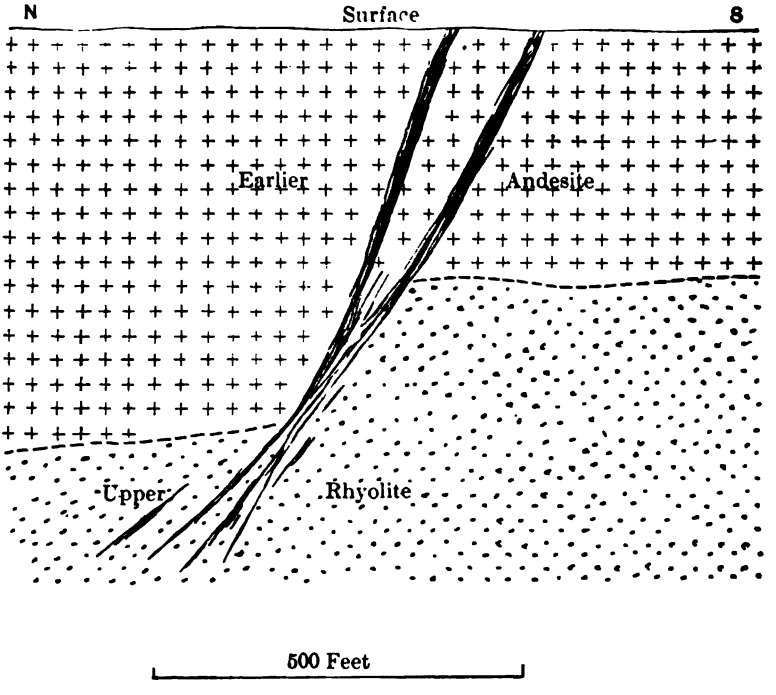


FIG. 2.—VERTICAL CROSS-SECTION SHOWING DOWNWARD EXTENSION OF TYPICAL VEIN INTO RHYOLITE.

To explain the superior productivity of the andesite, many hypotheses are possible: (1) The andesite is the earliest rock; the chief ore-mineralization followed it and preceded the other rocks. (Spurr's hypothesis.) (2) The source of the ore-minerals may have been the andesite itself or the upper rhyolite.

(Suggested by Burgess.) (3) The ore was deposited largely by metasomatism. The various rocks, particularly the upper rhyolite and the earlier andesite, present great contrasts in texture. Certain textural and chemical properties possessed by the andesite caused it to be more favorable to the precipitation of the ore-minerals than the other rocks. Or the andesite was more favorable to the formation of initial channels than the other rocks. (4) The path of travel of transporting-agents was mainly along the andesite-rhyolite contact and upward into the andesite. (5) Ore-deposition was a superficial phenomenon, effected through the decrease of heat and pressure near the surface, or through other superficial agencies. The andesite, at the time of ore-deposition, was the surface-rock, and, therefore, received the bulk of the ore-mineralization. (6) Post-vein faulting at the contact between upper rhyolite and lower andesite in some instances caused the disappearance of the vein at the contact. (7) Ore-mineralization occurred after the eruption of the earlier andesite and in cooling-shrinkage cracks in that rock.

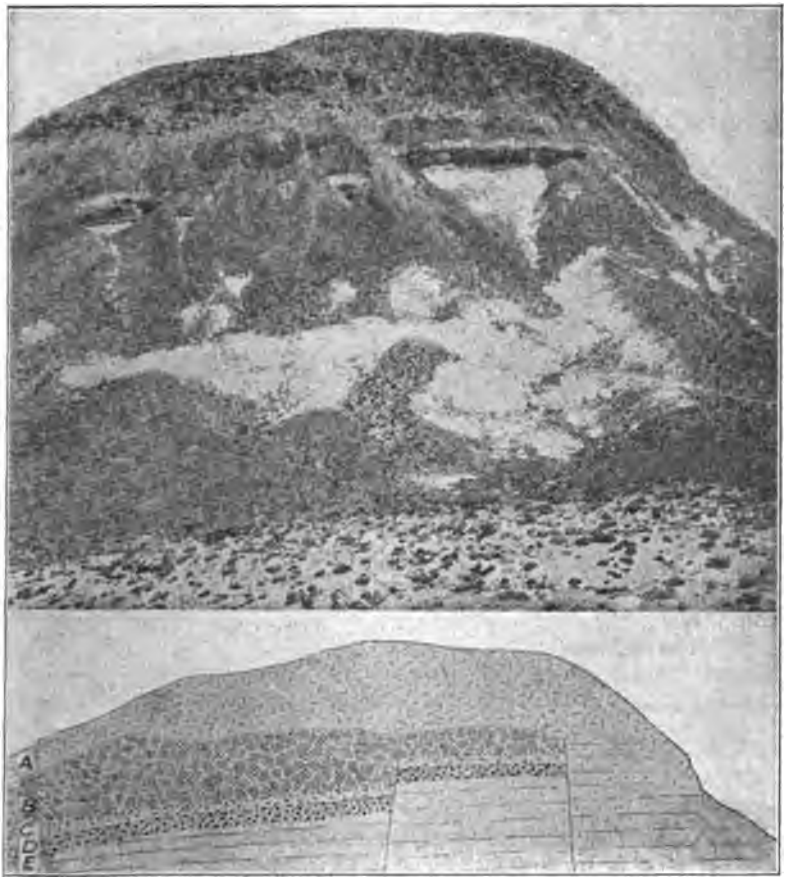
Spurr, as has been already made clear, accepts the first of these hypotheses and, apparently, rejects the others. The fact that the ore frequently extends down into the rhyolite, he explains by a supposition of several periods of mineralization, of which the first and most important was earlier, and the others later, than the rhyolite. Veins contained entirely in andesite belong, then, to the first period; veins in rhyolite, or in both rhyolite and andesite, belong to later periods.

That several periods of mineralization did exist is probably true. The assignment, however, of a particular vein to a particular period is often impossible. There is no mineralogical distinction whatever to be made between many veins which, according to the hypothesis, should belong to separate periods, and the sole apparent reason why one should assign them to separate periods is that, by so doing, he harmonizes the facts with the theory of rock-intrusion.

The remaining six hypotheses cannot be easily cast aside. To prove any one of them would be difficult; to disprove any one of them would be even more difficult. Yet so long as they stand as possible explanations of the localization of the

ore-deposits, they must offer impassable barriers to the acceptance of the hypothesis just now considered.

The Structure of Siebert Mountain.—Though it can have only an indirect bearing on the subject at hand, a consideration of



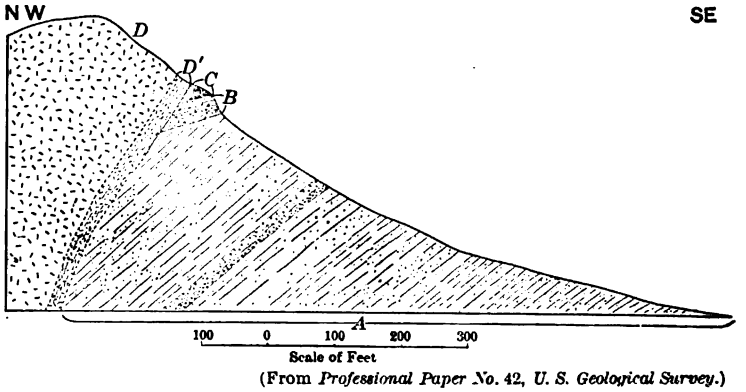
(From *Professional Paper No. 42, U. S. Geological Survey.*)

A. Later intrusive dacite.
B. Basalt.
C. Pumice agglomerate.
D. Basalt agglomerate.
E. White tuffs (lake-beds).

FIG. 3.—FACE OF SIEBERT MOUNTAIN, FROM THE SOUTHEAST.

the structure of Siebert mountain, situated several miles SW. of Tonopah, will illustrate in a specific instance the wide differences in opinion produced by the differences in interpretation here being discussed.

Siebert mountain looks like a succession of flows, Fig. 3. But Spurr's interpretation, shown in Fig. 4, regards it as partly intrusive. The fact of the matter is, however, that the supposedly intrusive rhyolite (or dacite), just above its contact with the basalt, has a very well-marked and nearly horizontal flow-structure—a fact scarcely compatible with the idea that the contact has the character shown in Spurr's section. Indeed, this idea would seem to be the result of *a priori* reasoning, were such a thing not manifestly impossible in the case of an investigator of Spurr's eminence and acumen.



(From *Professional Paper No. 42, U. S. Geological Survey.*)

A. Finely-stratified Siebert tuffs (lake-beds) with occasional layers of rounded pumice fragments or water-worn lava.

B. Basaltic agglomerate with bombs, capped by solid basalt.

C. Basalt.

D. Brougher dacite, intrusive neck.

D'. Glassy marginal facies of dacite.

FIG. 4.—VERTICAL CROSS-SECTION OF SE. SIDE OF SIEBERT MOUNTAIN.

Summary of Conclusions.—That the rocks of the productive part of the Tonopah mining-district are flows lying in the order of their deposition, is proved by the occurrence of volcanic tuffs at the contacts the interpretation of which has been in dispute. The other available evidence in some instances supports, and in no instance contradicts, this conclusion. There is no good reason, then, for the belief that the rocks underlying the earlier andesite are younger than the chief productive mineralization. Exploration in these rocks is accordingly relieved of the discouragement which would attend this belief.

The Laws of Igneous Emanation Pressure.

BY BLAMEY STEVENS, NEW YORK, N. Y.

(San Francisco Meeting, October, 1911.)

In this paper, which is a logical extension of my paper, The Laws of Intrusion,[1] the various pressures of emanation and their mechanical causes and effects on the large scale of nature are determined, and a classification of emanations is deduced therefrom.

The Subterranean Sea.

The subterranean sea is here defined as the intercommunicating water contained in the trains of pores and fissures of the rocks down to great depths. Whether the actual amount of water constituting this subterranean sea be large or small, does not enter into the present argument, since the static pressure of this water at any particular depth is not a function of the quantity: it involves only the depth and the density of the water.

Densities.

The total pressure at depths down to 10 or 15 miles is made up of the rock-stresses investigated in my previous papers, and the pressure of the subterranean sea. The latter component must evidently be equal to the weight of a solid column of the overlying water of unit sectional area. As water expands with heat, it should really be necessary to sum the density at every level. A knowledge of the densities of water under a variety of conditions is therefore needed, but does not seem ever to have been ascertained for such combined pressures and temperatures as we have to consider.

The specific gravity of water at 28° C. for various pressures up to 400 atmospheres, has been determined by Barus,[2] who

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[1] *Trans.*, xli., 650 (1910).

[2] *American Journal of Science*, Third Series, vol. xli., No. 242, p. 110 (Feb., 1891).

tried also to obtain results for higher temperatures, but was foiled in this attempt by the circumstance that the hot water rapidly dissolved the glass of the capillary tubes in which his experiments were conducted. No other systematic attempt seems to have been made since.

The specific gravity of water at 365° C. and 200 atmospheres (the critical temperature and pressure) has been determined as 0.429. At the normal temperature-increase of 1° C. per 100 ft. of depth, the above temperature corresponds to about 6.5 miles of depth; but the pressure would be obtained in the first 1.25 miles. It is impossible to say, without experimental research, to what extent the expanded water at specific gravity of 0.429 would be compressed by a pressure of more than five times the critical pressure. The judgment of nine out of ten scientists would, however, probably place the increased specific gravity somewhere near unity, *i.e.*, equivalent to the maximum gravity under surface-conditions. There are several reasons for this conclusion, but they will not be discussed in this paper. We shall adopt this figure as a tentative one for all depths.

We can with much more certainty assume a constant value for the specific gravity of rocks. We know that with the temperatures and pressures considered, this does not alter much. The volumetric expansion for a difference of temperature of 100° C. is only 0.0017 for marble, and 0.0024 for granite, while the volumetric compression is about the same for the corresponding head of 10,000 feet.

Pressures of Igneous Magnas.

According to my paper, The Laws of Intrusion, the pressure of an igneous magma is equal to the sum of the rock-pressure, its cohesive stress, and the pressure of the subterranean sea.

In order to arrive at a numerical measure of these pressures, let D be the density of the rock, and d that of the water. Taking H to be the depth, or head, and C the cohesive stress of the rock if broken by the magma, the total vertical magma-stress will be approximately $DH + C$. For equi-pressure country the horizontal stress is also $DH + C$, which is the magma-pressure of an irregular dike, sill, or laccolith. Except

under rare conditions, it is the maximum intrusive pressure possible.

In fissuring country DH remains the vertical pressure. Of this pressure the part dH is the pressure of the water, and the remainder (D — d)H must show the stress of the rock.

It has been shown[3] that the horizontal rock-head in fissuring country is given by $\frac{H}{I}$, where $I = \frac{1 + \sin \phi}{1 - \sin \phi}$, in which equation ϕ is the angle of the slip, tan ϕ being the coefficient of friction. The horizontal rock-pressure is therefore $(D - d)\frac{H}{I}$, and adding the water-pressure, the total horizontal pressure is $(D - d)\frac{H}{I} + dH$. This is the minimum magma-pressure possible.

Assuming the mean specific gravity of the rock to be 2.7, and the maximum value of I to be 5, the minimum pressure is approximately $\frac{DH}{2}$, or less than half of the maximum pressure.

Exhalations.

We may assume that, as molten magma rises, it frees itself more and more from occluded gases. These form what we may term "exhalations." Although they inevitably emanate first as gases, it must be apparent that after they leave the magma they are often cooled to the normal temperature of the surrounding rocks and mixed with the meteoric waters.

Residues.

If some of the occluded gases are still retained under external pressure while the magma is molten, they may be forcibly expelled from the magma as it consolidates. These may be termed "residual emanations" or "residues," since they form during the last stage of vulcanism and are left after the main magma-mass has consolidated.

Dry Magmas.

When all the water in a magma has been exhaled it may be called dry. In this condition it still contains the remainder of those soluble substances which are not exhaled but might have

[3] The Laws of Fissures, *Trans.*, xl., 475 (1909). This particular formula for I may be easily deduced from those given.

been residues had any water been left for their solution. These may include the uncombined forms of gold, silver, copper, etc. The Lake Superior copper-deposits are possible examples. It has never been shown, however, that the whole of the sulphur in copper-bearing magma could be exhaled.

Intrusive Emanation.

An intrusive emanation may be defined as one which breaks a way for itself in the rocks which it traverses. It thus resembles in many respects an intrusion of magma (see Figs. 3, 4, 5, 6, 7, 8, and 10). In Figs. 4, 6, and 10 the emanation is considered to have opened the fissures, dikes, and sills, and wholly or partly deposited its minerals, before the magma which afterwards filled some of these cavities had reached them. In Fig. 7 the magma was deposited first.

Free Emanation.

A free emanation may be defined as one which exists in some cavity, crevice or other opening which has been formed previous to the intrusion of magma from which the emanation is derived. Most fissure-veins are of this type (see Figs. 13 and 14).

Pressure of Exhalation.

The pressure of exhalation at the point of emanation is essentially that of the magmas. This pressure has been found above and need not be here repeated.

Effective Exhalation-Pressure.

The pressure which is effective in producing a flow of exhalation through the fissures and cavities of the rocks, is the difference between the exhalation-pressure and that of the subterranean sea, statically considered, for the same depth. The effective pressures at the points of emanation are therefore:

$$(D - d)\,H + C \quad \text{(For equi-pressure country).}$$

$$(D - d)\,\frac{H}{I} \quad \text{(For fissuring country).}$$

These, it will be noticed, are the total rock-stresses—a result which might have been foreseen. In terms of water-head, these stresses are:

$$1.7\,H + C \quad \text{(For equi-pressure country).}$$

$$0.34\,H \quad \text{(For fissuring country).}$$

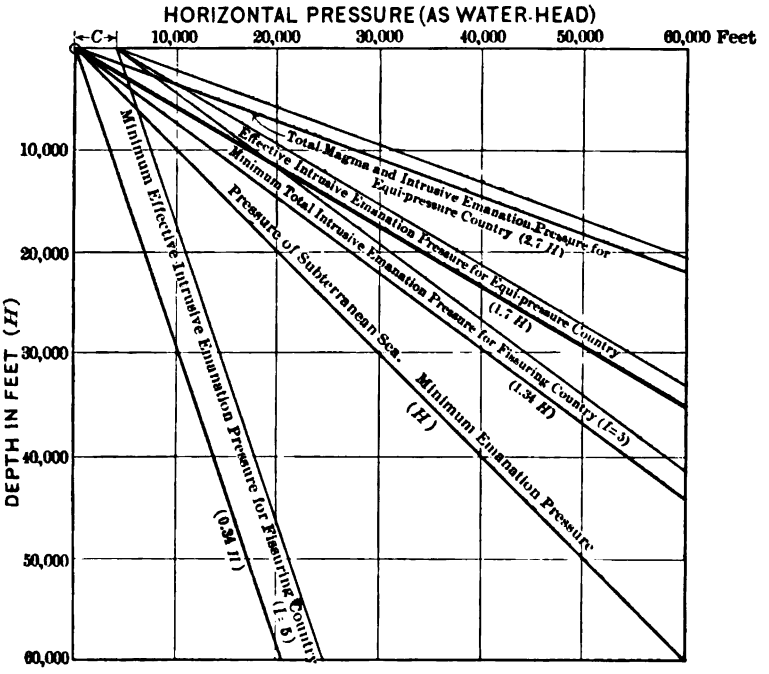

FIG. 1.—DIAGRAM SHOWING VARIOUS LAWS WHICH GOVERN EMANATION-PRESSURE.

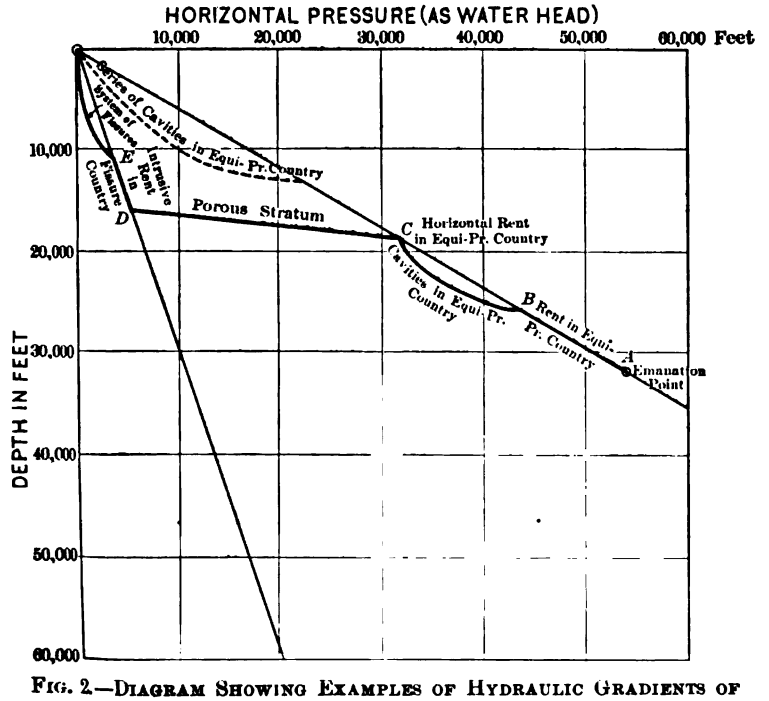

FIG. 2.—DIAGRAM SHOWING EXAMPLES OF HYDRAULIC GRADIENTS OF EMANATIONS.

FIG. 3.—QUARTZ DIKE, AURORA, ALASKA.
(From a photograph by the author.)

Fig. 1 shows the total heads and the effective intrusive hydraulic heads for equi-pressure country, assuming C as 4,000 ft. It shows also the minimum heads for fissuring country which is being intruded. The common effective head (1.7 H) applies to fissures, joints, and cleavage in equi-pressure country which is being intruded. This is called common and accentuated with a very thick line, because it represents a very usual condition of emanations.

Pressures and Effective Pressures of Residues.

The pressures and effective pressures of residues are limited by the rock-stresses; for we have reason to believe that expansion takes place on the final separation of the magma and residue. The maximum pressures and effective pressures are therefore the same as for exhalations. The minimum pressure is that of the subterranean sea, and the minimum effective pressure is *nil.*

Regional Pressures.

A system of fissures and cavities may become filled with liquid at a pressure approaching that of emanation. The porous rocks which are included in and surrounded by such a system are not subjected to any difference of pressure tending to cause flow within the pores. The deposition of mineral is therefore confined to the fissures and cavities. The mode of deposition is very different with a forced capillary flow of emanation, for in this case the pressures are not distributed over a region, but fall very rapidly.

Intrusive Emanation-Zones.

There is a good deal of difference between the intrusion of emanations and the intrusion of magmas. As we have seen, with emanations the common effective pressure is $(D - d)H$, and with magmas there is no effective pressure. The reason is that the emanation forms part of the subterranean sea, whereas the magma does not. The pressure of the water in the rocks immediately surrounding an emanation rises and falls with the pressure of the emanation, whereas the pressure of an intrusive magma is not so intimately connected with the water-pressure.

FIG. 4.—DIAGRAM SHOWING OCCURRENCE OF IRON-ORE WITH DIKES OF DECOMPOSED DIABASE AND DIORITE, MARQUETTE, MICH.

After Van Hise, *Monograph XXVIII., U. S. Geological Survey* (1897).

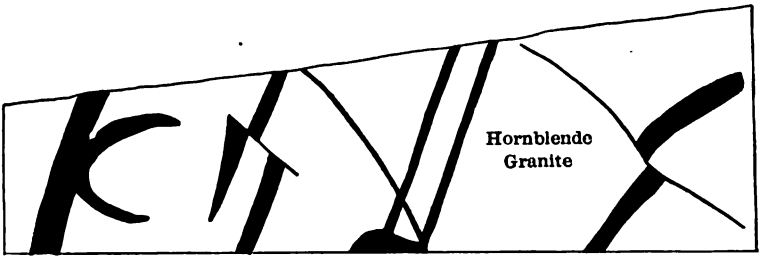

FIG. 5.—INTRUSIVE TIN-BEARING PEGMATITE "VEINS" IN HORNBLENDE-GRANITE NEAR OSHOEK, SOUTH AFRICA.

After W. R. Rumbold, *Trans.*, xxxix., 787 (1908).

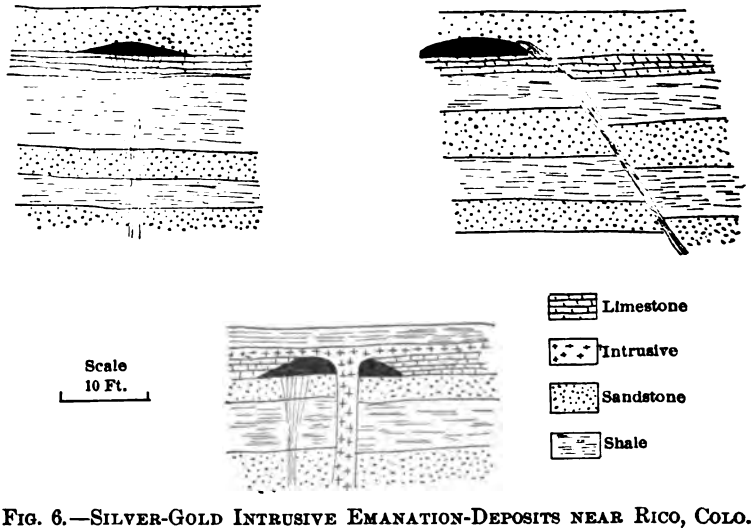

FIG. 6.—SILVER-GOLD INTRUSIVE EMANATION-DEPOSITS NEAR RICO, COLO.

After J. B. Farish, *Proceedings of the Colorado Scientific Society*, vol. iv., p. 164 (1892–93).

With the magma the rending-pressure is applied at one particular edge and the rending goes on along a definite plane. With the emanation, it is applied over a zone which may contain many slips, joints, and cleavage-planes, any or all of which may open up to some extent. This zone may be limited by physical conditions, such as the walls of a stronger or more impervious rock, or one with the cleavage in a different direction, or from many other causes. The zone affected may often take the form illustrated in Fig. 9, taken from the Juneau gold-belt of southeastern Alaska.

Quartz dikes may sometimes be found where harder or more compact rocks are encountered, *e. g.*, that shown in Fig. 3, from southern Alaska. The Leadville exhalations (Fig. 10) are partly intrusive; and the Independence mine of Cripple Creek (Fig. 7) is a fine example of intrusive emanation.

Subterranean Lakes.

Subterranean lakes may be defined as bodies of stagnant water isolated from the subterranean sea by impervious rocks.

Such lakes are common, become more frequent with depth, and may contain considerable saline matter. This may have been derived from the ocean or a salt lake at the time of deposition of the rock, by subsequent circulation of saline waters, by solution of saline deposits, from emanations, or by some combination of these sources.

The pressure of subterranean lakes is never below that of the subterranean sea, and is usually greater. This circumstance may be explained either by its connection with a source of igneous emanation or by the heating of the imprisoned waters above their original temperature. In the great majority of cases, however, it is doubtless due mainly to the slow squeezing of the containing rocks, either horizontally or vertically, so as to diminish the pore-space.

The limit to the pressures obtained is fixed by the breaking of the impervious boundary of the lake. The water-pressure required for this is the same as for an emanation. This similarity may sometimes make it hard to tell whether some intrusive solutions are of igneous origin or come from subterranean lakes, which have been subjected to stress. The absence of volcanic rocks may, however, be a fair criterion. Examples are

the lead- and zinc-deposits of the Mississippi valley and the massive salt-deposits[4] of the Louisiana coast. The latter deposits occur in horizontally-stratified rocks which are pushed up around

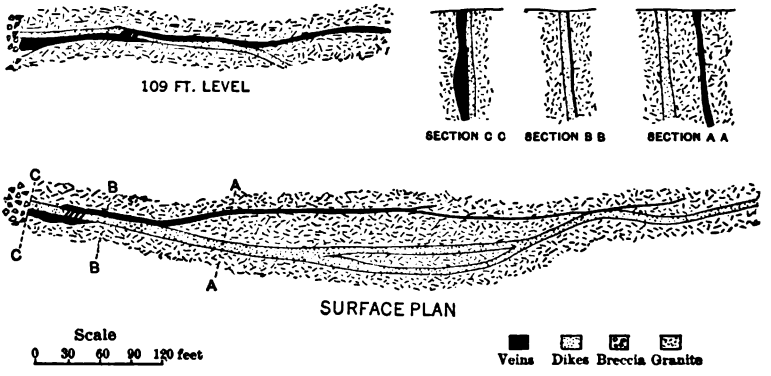

FIG. 7.—INDEPENDENCE MINE, CRIPPLE CREEK, COLO.
After R. A. F. Penrose, Jr., *Sixteenth Annual Report, U. S. Geological Survey*, Part II., p. 200 (1894–95).

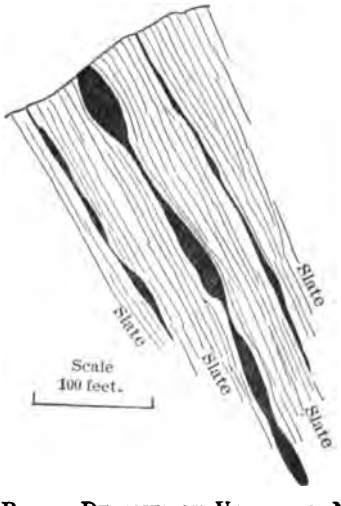

FIG. 8.—PYRITE-DEPOSITS OF VARALDSO, NORWAY.
After Vogt.

the edges of the deposit. In both cases the pressures concerned are evidently of intrusive magnitude.

[4] *Bulletins Nos.* 5 and 7, *Geological Survey of Louisiana* (1907, 1908).

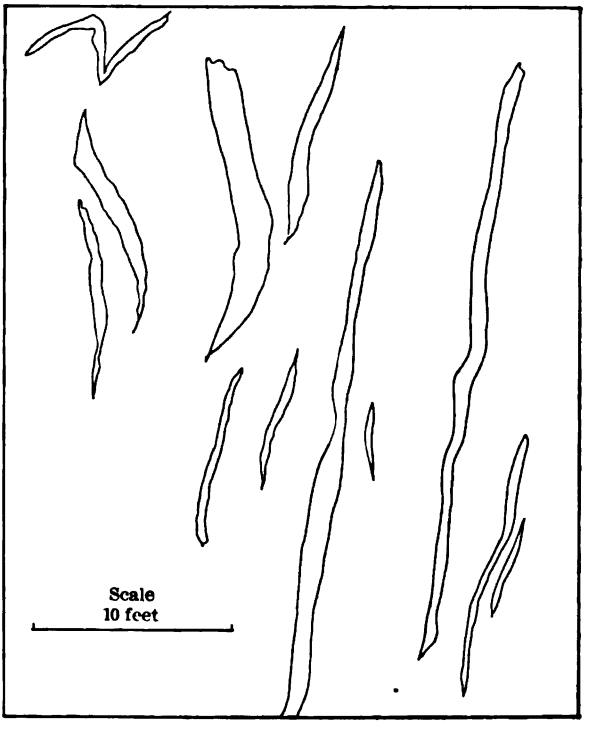

FIG. 9.—INTRUSIVE RESIDUAL GOLD-QUARTZ IN BROWN DIORITE, JUNEAU, ALASKA.

After Spencer, *Bulletin No.* 287, *U. S. Geological Survey* (1906).

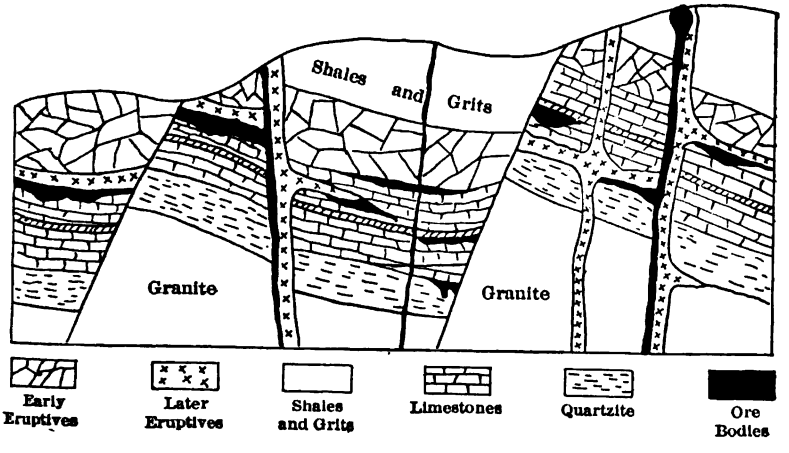

FIG. 10.—IDEAL SECTION OF BREECE HILL, LEADVILLE, COLO.

After Max Boehmer, *Trans.*, xli., 165 (1910).

Intrusive Residues.

There is strong evidence that residues may have a pressure high enough to form intrusions. I shall show in a later paper that they are the only ones which can contain large quantities of silica when they leave the magma. Intrusive dikes and lenses of quartz, etc., will therefore point to the intrusive pressures of residues. (See Figs. 3, 5, and 9.) It is of course possible that in some of these cases the silica was picked up after the emanation left the magma.

Explosive Emanations.

There are cases where intrusive rents are formed of such size that the emanations rapidly escape to the surface. They then lose little heat and expand freely, and so rapidly that the condensing water has no time to fill the rent. The effective pressures become equal to the whole pressure at the point of emanation, and may in some cases be so suddenly released to the atmosphere, or at comparatively small depths, that what we term a volcanic explosion takes place.

Flow of Igneous Emanations.

If the flow of emanations from magma goes towards the surface of the earth in a single straight path, with similar conditions along its entire length, the straight lines of Fig. 2 may be considered as the mean hydraulic gradients. The effective pressure at every place in the path of flow is then just equal to the rock-stresses. Any additional stress would further open the crevices of the rock. If, however, as must generally be the case, the flow covers a greater area, and encounters fewer obstacles, as it gets nearer the surface, the pressure-curves will be somewhat as represented by the dotted line in Fig. 2. The pressures at the two terminals remain the same, but the variation of pressure with depth decreases as the emanation gets nearer the surface. The dotted line shows the effective pressure to be less than the prevalent rock-stresses for the corresponding depth.

On the other hand, it is conceivable that a less porous stratum or bar of unfissured rock might be encountered between the source of emanation and the surface. At such places, the effective pressure would become greater than the corresponding rock-stresses, and the impervious rock would

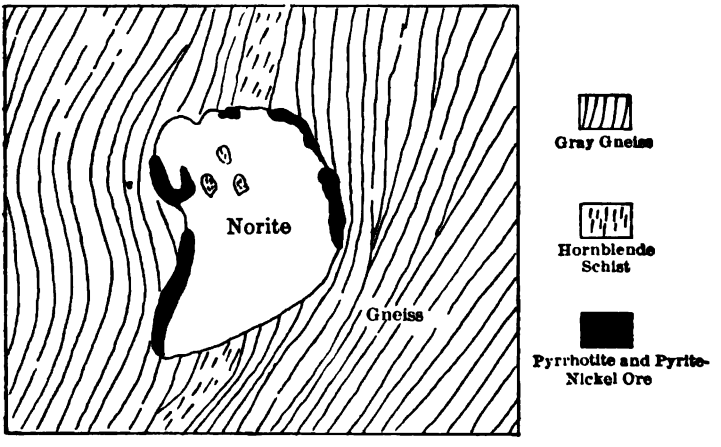

FIG. 11.—SKETCH-MAP OF MEINKJAR PYRRHOTITE- AND PYRITE-NICKEL FIELD.

After Vogt. *Zeitschrift für praktische Geologie*, vol. i. (1893). See *Nature of Ore-Deposits*, by Beck and Weed.

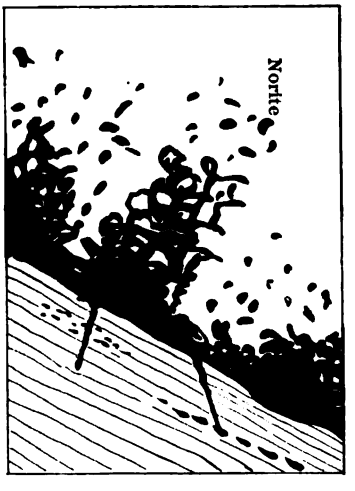

FIG. 12.—ENLARGED SECTION OF A PART OF NORITE-CONTACT SHOWN IN FIG. 11.

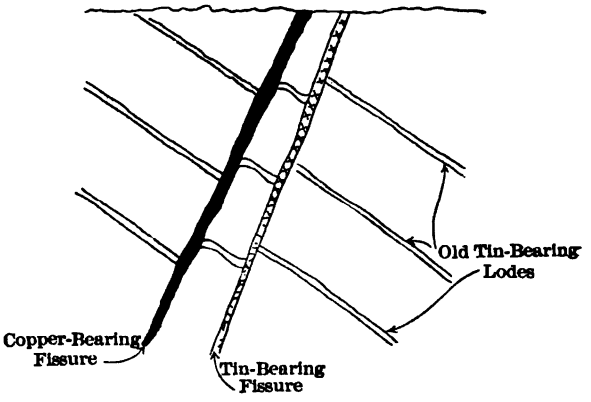

FIG. 13.—SECTION OF SEAHOLE MINE, CORNWALL, ENGLAND.

After Carne. See *Geology*, Prestwich, vol. i., p. 318 (1886).

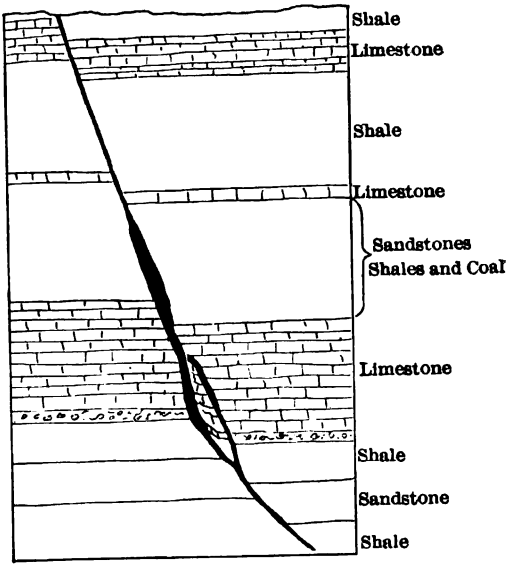

FIG. 14.—SECTION OF A LEAD-VEIN, ALSTON MOOR, CUMBERLAND, ENGLAND.

After Wallace. See *Geology*, Prestwich, vol. i., p. 320 (1886).

rupture, until conditions of equilibrium were again established by the rending of channels of sufficient size.

The heavy lines of Fig. 2 illustrate a hypothetical case. Starting at *A*, the point of emanation, the intrusive pressure rends a path or series of paths for itself until it reaches *B*. There the emanation finds a series of previously-formed cavities and travels by them until it reaches *C*, which is the under side of a porous bed, where the cavities cease or are choked. The emanation then rends a horizontal path for itself along the bedding and thereby exposes enough area of porous material to permit it to flow upward through the bed, with rapid loss of pressure, to *D*, the lowest part of the country affected by normal fissuring-pressure. It has, however, to rend its way to *E*, before it strikes the series of normal fissures by which it finally reaches the surface at 0.

The lead- and zinc-deposits of the Mississippi valley are probably good examples of discontinuous rupture.[5] W. P. Jenney, who examined this field for the U. S. Geological Survey, says that, although not associated with visible intrusions of igneous rocks, all the lead- or lead-and-zinc regions of the Mississippi valley occur along three lines of upheaval, and that the vertical fissures are the channels through which the mineral-bearing solutions have entered the rocks. These so-called fissures have in most cases been almost completely closed again; and the absence of displacement has been the chief argument against their having been carriers of mineralizing solutions.

Capillary Flow.

Capillary flow often starts from the point of emanation. This takes place at an igneous contact where there is no other means of egress for emanation (see Figs. 11 and 12).

In other cases the emanation may be conveyed to the porous medium by a fissure or intrusive rent (see Figs. 6 and 10). A number of lead- and zinc-deposits come under this head. Usually the porous medium serves to distribute the hydraulic pressure over a larger area, so that no further fissures or rents are required to convey it. The porous material also greatly reduces the effective pressure and robs the contents of every

[5] *Trans.*, xxii., 171 (1893).

mineral which can possibly be replaced under existing conditions of pressure and temperature. The effective emanation-pressure within a few inches or feet of the magma may under particular circumstances be *nil.*

Classification of Emanations.

A classification of emanations is given in Table I. This table is constructed on the same plan as that proposed for fissures and intrusions in my previous papers; that is to say, an emanation of either class may be of either type, and each type of class may have either form of flow, so that there are eight different divisions, into one of which an emanation may fall. This, be it remembered, is without considering the change in character of the emanation after it leaves the magma. The class is fixed from the derivation, and not the actual condition of the emanation at the time of mineral deposition. Of the eight divisions, there are only six which need be considered, the intrusive capillary combinations being rare and unimportant.

It must be again noted that the question whether the ore-bearing fluid is gaseous or liquid at the time and place of deposition does not enter into the present discussion, which considers, first, the conditions at the time and place of emanation; secondly, the pressure at the time and place of deposition, and, lastly, the rate of change of pressure at the time and place of deposition. These leave their marks respectively on the deposit: first, in its chemical constitution; secondly, in the shape and position of the cavity filled; and lastly, in the size of the cavity or system of cavities.

TABLE I.—*Classification of Emanations.*

	Class.	Derived from.	*Type.*	Cavity.	Effective Pressure.	*Flow.*	Size of Channel or Cavity.	Rate of Variation of Effective Pressure.
Igneous Emanations. (Gases and liquids expelled from igneous magmas.)	*Exhalations*...	Liquid magmas.	*Free*....	Not ruptured or opened by emanation-pressure.	Less than least rock-stress.	*Capillary*.....	Small.	Rapid.
	Residues.....	Solidified magmas.	*Intrusive*.....	Rent or ruptured by emanation-pressure.	Greater than least rock-stress	*Super-Capillary*.....	Large.	Slow.

Physical Data of Igneous Emanation.

BY BLAMEY STEVENS, NEW YORK, N. Y.

(San Francisco Meeting, October, 1911.)

My previous paper is entitled, The Laws of Igneous Emanation Pressure. The present paper lays no claim to the exactitude and completeness of a law, since it is of a provisional nature and may be disregarded when experiments have established a basis for more exact reasoning.

The general physical states in which aqueous emanations may occur are first considered. The possible physical conditions under which the mineral elements may occur in emanations are then discussed, and a classification of deposits, dependent on emanation, is proposed.

A system of scientific experimental investigation is outlined, and it is shown that the physical and chemical properties of emanation may be exactly determined thereby. The investigation is urged as feasible and simple, and not necessarily requiring large and costly apparatus. It is the only step necessary to make possible the formulation of a complete set of laws of igneous emanation.

Process of Emanation.

In a rising column, magma ranges in temperature from 900° to 1,200°C. Exhalation must begin at some fairly great depth. What this depth may be we can only surmise, since no experimental evidence is available. Exhalation, however, continues as the magma flows upward, until at another level the occluded gases have all escaped. This depth is probably upward of 5,000 ft. (See Fig. 1.)

All exhalations must come from between these two levels. As the whole emanation is the sum of the exhalations and residues, the residues must come from below the uppermost level. They will be most abundant when coming from those parts of the intrusion which have solidified below the lower level.

Conditions at Time of Emanations.

The physical and chemical conditions of emanation are as yet little known. Barus[1] has shown that colloidal silica is miscible in all proportions with water, at all temperatures and pressures at which water can exist as a liquid. It seems, however, certain that the universal tendency is for the colloidal to

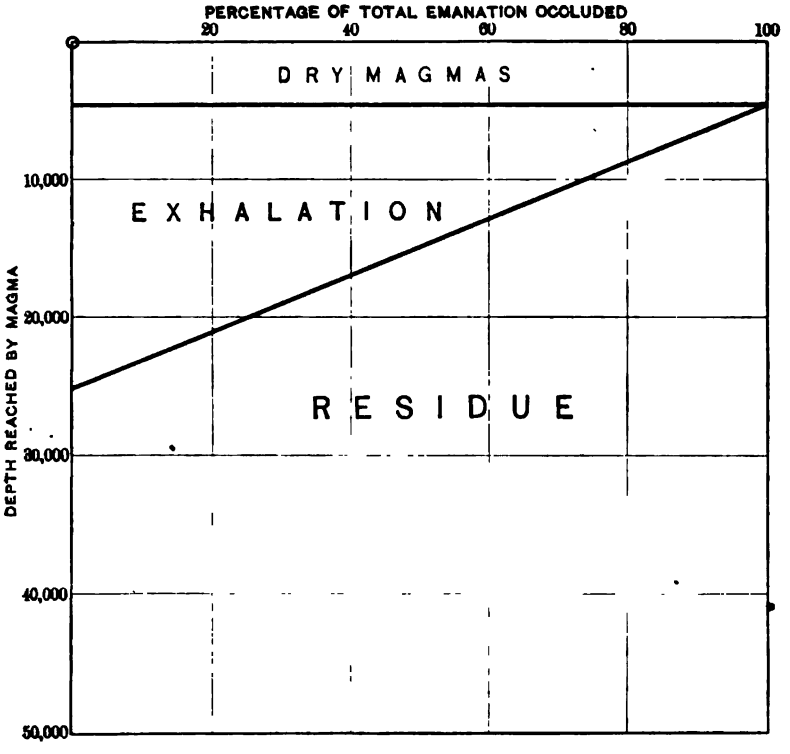

FIG. 1.—DIAGRAM SHOWING RELATIONS OF THE TOTAL EXHALATIONS AND RESIDUES OCCLUDED FROM A MAGMA WHEN IT HAS REACHED VARIOUS DEPTHS. (Actual depths of percentages are merely speculative guesses, for an average magma, there being no experimental data.)

assume the crystalline form; and no solution of crystalline silica has ever been made in pure water. Barus tried, unsuccessfully, up to 600°C.,[2] the rock and water being heated in iron

[1] *American Journal of Science*, Fourth Series, vol. vi., No. 33, p. 270 (Sept., 1898). See also C. R. Van Hise, *Sixteenth Annual Report, U. S. Geological Survey*, pt. I., p. 687 (1894–95). In this publication Van Hise first states this important principle. Barus (*ibid.*) states that the melting-point of aqueo-igneous silica frequently lies below 200° C.

[2] *Fourteenth Annual Report, U. S. Geological Survey*, pt. I., pp. 161–162 (1892–93).

tubes capable of withstanding great pressure. As liquid magma contains colloid silica, there can be little doubt that water will mix with it, if the pressure is great enough to retain the water in a moderately dense fluid form at these great temperatures. There is, however, evidently a limit to the miscibility of steam and magma.

Miscibility of Liquids.

It must be conceded that molten silicates are as completely miscible in depth as on the surface of the earth. The general trend of physico-chemical theory and experience leaves little doubt of this fact, and no forcible evidence to the contrary has been discovered.

There are also good grounds for the presumption, stated above, that liquid silicates are completely miscible with liquid water.

Distinction Between Liquids and Gases.

Above the critical temperature, there is no sudden change of density separating the liquid and the gaseous forms of matter. The same change of properties must take place, however, though it may not be at any well-defined temperature.

The distinction between liquid and gaseous conditions may be determined from one or more of these changes of property; for example, all exhalations are presumably gaseous when formed.

The distinction between vapor and liquid at great depths may be most conveniently prescribed and defined by the miscibility of the fluid with liquid silicate magma. The liquid may be presumed to be completely miscible with the magma, but the vapor to mix only as an occluded gas; that is to say, in such amounts as its pressure may limit or determine.

The pressures under which the different forms of emanation may exist can be properly determined by experiment only.

Up to the critical point there is a density-criterion to separate the liquid and gaseous states. The specific gravity of water at the critical temperature (365°C.) and pressure (200 atmospheres) is 0.429. This density might possibly be the criterion separating the liquid and gaseous states over a considerable range of temperature-pressure conditions. If this is the case at the temperature of the magma (from 900° to 1,200°C.), a

depth of several miles could be obtained at this temperature before the gaseous condition of water would disappear. We should probably be not far off in assuming that exhalations are so formed at five miles in depth. This is a much greater depth than has hitherto been considered as possible with exhalations; hence we have a new phase of ore-genesis to consider.

General Effect of Temperature and Pressure on Emanations.

Many effects of temperature and pressure on gases are amenable to simple laws. In the case of liquids, however, they are much more complex. Increase of temperature usually increases the solubility of substances; there are a few exceptions. Increase of pressure on solutions is not important. Increase of temperature increases the vapor-density or vapor-pressure of any substance. In dealing with the constituents of an emanation we may be aided by the fact that those which cannot be exhalations must of necessity be residues, and *vice versa.*

Effect of Temperature and Pressure on Exhalations.

The effect of temperature and pressure on gaseous substances may be fairly well estimated.

Every liquid, at every temperature, must be in equilibrium with its own vapor. The vapor-pressure is practically the same for all external pressures. As all gases are perfectly miscible, every gas in contact with a liquid must absorb some of it.

The weight of vapor absorbed per unit-volume is independent of the pressure of the gas, or approximately so. Suppose a liquid has a vapor-pressure of one atmosphere and a gaseous solution at 1,000 atmospheres pressure absorbs all it can from the liquid. The gas will then have one-tenth of 1 per cent. by volume of the vapor. If the gas is that of water, and the liquid that of some metal or mineral, the density of the mineral vapor will be much greater than that of the water, and the percentage by weight will be much more than one-tenth. If the water is a vapor itself, instead of a perfect gas, the quantity will be somewhat different.

It is not the intention of this paper to arrive at exact figures, however, but only to show the right perspective of the great forces involved.

The following is a list of vapor-temperatures at atmospheric

pressure for a few substances for which they have been determined:

Table of Boiling-Points.

	Degrees C.
Mercury,	350
Selenium,	700
Zinc,	960
Lead,	1,040

Since most of these temperatures are a good deal below that of average liquid magma, and vapor-pressures rise in exponential ratio, it is reasonable to suppose that at magma-temperature the vapor-pressures would be many atmospheres.

A pressure of 1,000 atmospheres corresponds to an emanation of equi-pressure country at about 13,000 ft. of depth, which is perhaps an average depth for exhalation. In general, therefore, we see that the substances tabulated, and many others which we know to be at least as volatile, may be contained in exhalations in sufficient quantity to account for ore-deposits.

It is, of course, not to be presumed that the exhalations remained gaseous, but more often that before they deposited their mineral they became liquid.

Residues.

Reasoning along the same lines, it appears that certain elemental solutes which we know to be contained in emanations must be confined to the residues.

The volatile properties of the real compound substances which enter into emanations have not been determined; as a matter of fact, the exact combinations are not properly known. We know, however, that in general, compounds are less volatile than their constituent metallic elements. The following common ore-minerals are worthy of special discussion:

Silica.—It is extremely unlikely that silica can be a gas under igneous conditions; hence, we may conclude that silica does not occur in exhalations freshly formed. It is quite likely, however, that exhalations may become condensed to the liquid form after they leave the hot magma, and that they may then dissolve colloid silica from the solidified glassy portions of surrounding rocks, or, that some more base constituent of the emanation may replace and take away the silica of other minerals in the rocks it penetrates.

The same argument does not apply to residues, for these may be of liquid form when segregated from the parent magma. As far as my reading and observation go, these conclusions have been borne out. Where pegmatite and more acid deposits are associated with volcanic rocks, there is often evidence that they are the products of expiring vulcanism.

The emanation from basic rocks does not seem to contain much silica. Apparently, therefore, it is only the excess of silica which goes off as emanation. Any silica which can combine with the oxides of the metallic elements to form the less soluble silicates remains in the rocks.

Water is probably given off at greater pressures when there is little or no excess of free silica. This follows from the known fact that colloidal silica shows some affinity for water, an effect which is increased by pressure, there being considerable diminution of volume when colloid silica is dissolved in water.[3]

Silica-Minerals.—The typical silicates, such as those of soda, potash, lime, magnesia, alumina, etc., are presumably held in combination with the silica and do not go off with the aqueous vapors; hence, like silica, when occurring in emanations, they are mainly residues.

Lead, Zinc, Etc.—Lead, zinc, and possibly some copper and other easily volatilized metals, are usually found in the primary condition as sulphides. They may undoubtedly leave the magma as exhalations.

Vapors.

Between the liquid and the gaseous phases of matter there is an intermediate or vaporous phase in which intermediate properties may exist. This vaporous phase is scarcely noticeable at ordinary pressures, but becomes more important when the pressure is such as to give the vapor a density approaching that of the liquid phase.

Many substances may thus be associated with vaporous exhalations which are not volatile in themselves.

Gold and Silver.—Gold occurs in emanations in such small quantities that the general rules applicable to the predominating minerals cannot be applied.

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[3] Barus, in *American Journal of Science*, Fourth Series, vol. x., No. 5, p. 173 (Feb., 1900).

Gold and silver may be slightly soluble in vapors of tellurium, etc.; silver in lead, zinc, antimony, etc.

Iron, Copper, Nickel, Etc.—Iron occurs primarily either as one of two sulphides or one of two oxides.

Deposits of pyrite and pyrrhotite are sometimes very massive and free from silica. They contain sulphides of copper, and possibly of nickel, which are evidently derived from the same emanation. They are no doubt often derived from vaporous exhalations.

Tin.—Tin occurs primarily as an oxide, which is only fusible with great difficulty. In all likelihood, it comes from residues.

Minerals Indicating Probable Class of Emanation.

Residues.	Residues and Exhalations.	Exhalations.
Silicas.	Iron Sulphides.	Galena.
Silicates.	Copper Sulphides.	Sphalerite.
Cassiterite.	Nickel Sulphides.	Stibnite.
	Gold- and Silver-minerals.	Realgar.
		Cinnabar.
		Tellurides?

Former Theory and Nomenclature.

The division of emanations into exhalations and residues is not new; but it has not been previously recognized that there is a considerable range of depth at which these two forms may both emanate from the same magma at somewhat different periods of its history.

Residues have been heretofore called pneumatolytic,[4] under the misapprehension that they are gaseous. On the contrary, we have shown that residues have more of the properties of liquids than of gases, so that the name pneumatolytic is out of place.

Exhalations have previously been considered only as belonging to very small depths, and have been called fumarolic. It has apparently been presumed that there was no emanation between the deep-seated "pneumatolytic" and the very shallow "fumarolic" emanations.

The old theory that all good veins ought to be fissures is also losing ground. Many of the most famous mines of the world will probably be found to be intrusive rents. These include nearly all the large stock-works, many flat deposits,

[4] See *Ore-Deposits*, a Discussion; edited by T. A. Rickard.

such as those of Leadville, and most vertical deposits which show no signs of lateral displacement.

It may be necessary to note here that the present paper does not concern itself with the causes of deposition of mineral from the emanations; it eliminates as far as possible all chemical phenomena; and its terms in this respect must be taken as general. Thus, for example, in speaking of the solubility of sulphides, it is not to be understood that the sulphides necessarily go into solution as such.

Indirect Effects of Emanation.

The water contained in magmas may have considerable effect on the crystallization of rocks. Thus, the coarse crystallization of deep-seated granite may not be altogether due to slow cooling. It may, to a very large extent, result from the reduction in viscosity, due to included water. Rocks of similar composition, and cooled just as slowly, but near the surface, do not contain this water, and consequently form fine-grained or even glassy varieties.

Other modes of demonstrating the presence or absence of water during crystallization are at least to be expected: for example, the determination of the exact minerals or proportions of minerals formed by the crystallization of a magma of given composition, *e. g.*, orthoclase and microcline, the latter being formed from the more aqueous solution. It may be presumed that microcline is orthoclase on a greatly magnified scale, this larger crystallization being due to the reduced vicosity of the fluid. Dana[5] says that the essential identity of orthoclase and microcline has been urged by Mallard and Michel-Lévy on the ground that the properties of orthoclase belong to an aggregate of sub-microscopic twinning lamellæ of microcline, according to the albite and pericline laws.

Investigation may show that the amphiboles and pyroxenes are related in some similar manner. The manner of distribution of iron as between higher and lower oxides and sulphides is also open to study.

Experimental Possibilities.

The pressures involved in emanation are not too high to be amenable to experiment. Fire-arms are frequently called upon

[5] *System of Mineralogy*, 6th ed., p. 323.

to withstand an internal gaseous pressure of 50 tons per square inch, which corresponds approximately to 16 miles of depth under rock-head. High-speed steel is now manufactured which has great strength up to a red heat, or 700° C.

The water and other necessary ingredients for a series of experiments could be placed in a tube bored out of this steel. This could be lined with platinum and sealed by calking the

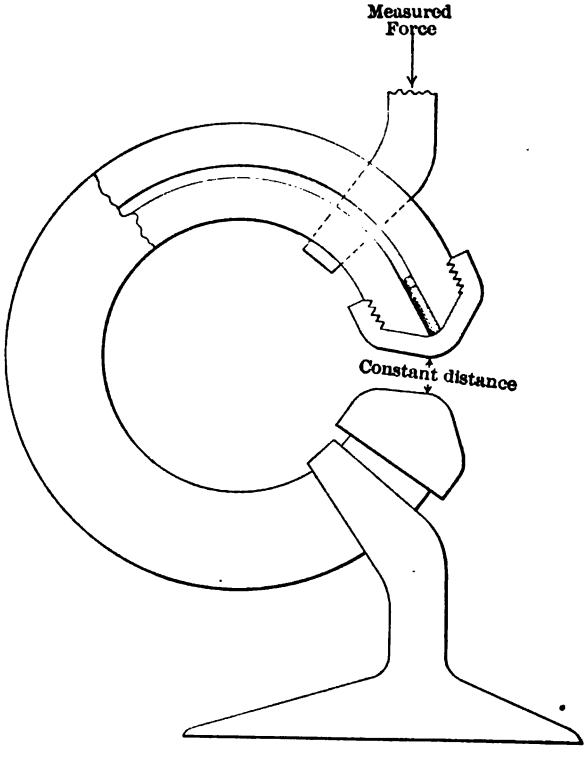


FIG. 2.—SUGGESTED EXPERIMENTAL APPARATUS FOR THE DETERMINATION OF THE PROPERTIES OF EMANATIONS AT MEASURED TEMPERATURES AND PRESSURES.

open end with a platinum plug held in place by a steel screw-cap. If the tube were sealed while the water was kept boiling no air would remain in the tube, and the exact amount of water could be determined by weighing after the tube was sealed. The pressure could be measured by having the tube bent into a circle so that the two ends nearly meet. The distance between the two ends is then an index of the pressure within. The change

of elasticity of the metal with temperature might have to be reckoned with; but it may be practically eliminated by having the distance between the two ends always brought to a constant amount by the imposition of an external stress. This could be accomplished with weights as shown in Fig. 2, the weights required bearing a fixed proportion to the internal pressure. The value of this factor might be verified by connecting the apparatus to as large a known hydrostatic pressure as possible.

If the temperature of the charged sealed tube were gradually raised, and the pressure observed at frequent temperature-intervals, a series of pressure-temperature equivalents of saturated steam would be obtained up to some point below the critical temperature, where the contents of the tube would be all liquid or all gaseous. Above this temperature the density would also be known, as it would remain approximately constant for any series of pressure-temperature observations. Different weights of water could be taken in several different series of pressure-temperature observations.

Every conceivable condition of fluid, as regards its pressure, temperature, and density, might thus be exactly co-ordinated between the two limiting conditions, namely, that of a density or pressure too small to be measured, and that of a pressure or temperature too high for the apparatus to withstand.

Besides the physical properties of water under high temperatures and pressures, there are, of course, to be studied the physical relations of an almost infinite number of solutions and mixtures, which are open to experiment with the same apparatus. There are also numerous chemical reactions to be elucidated. Should not some such experimental investigation be undertaken? The economic importance of the subject seems to me fully to justify it.

Every advance which leads to the more exact classification of mineral deposits limits the need of speculation in mining, gives more basis for local investigation, and in the aggregate lends itself to the saving of wealth. This is the *raison d'être* of the mining engineer.

Gold-Mines in Southern Colombia.

BY FORTUNATO PEREIRA GAMBA, TUQUERRES, COLOMBIA, S. A.

(San Francisco Meeting, October, 1911.)

I. General Description.

The Department of Nariño (formerly included in the Department of Cauca) is a narrow band about 170 km. (100 miles) wide in the southern part of the Republic of Colombia,

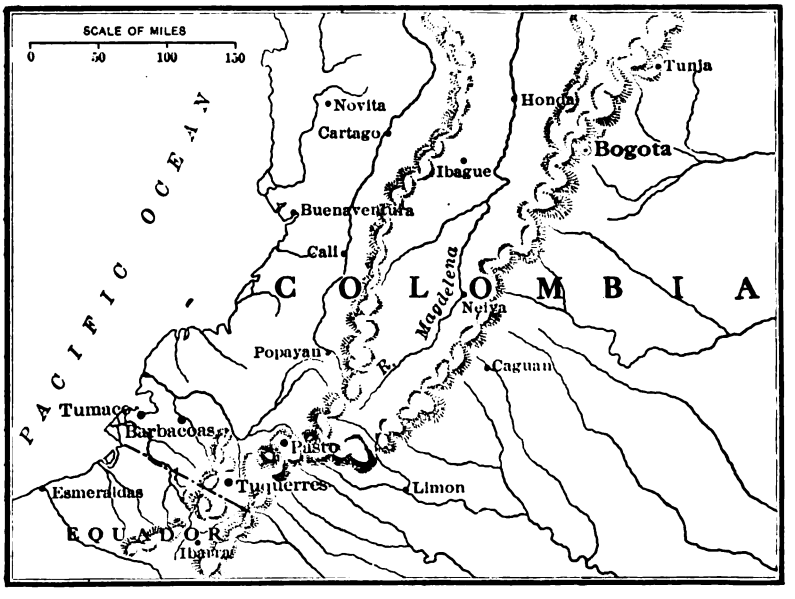


Fig. 1.—Sketch-Map of the Southwestern Part of Colombia.

near the boundary of Ecuador, as shown in the sketch-map, Fig. 1. Its capital is Pasto, a small city of 25,000 inhabitants; the principal towns are Túquerres, Cumbal, Ypiales, Barbacoas, and Tumaco. The port on the Pacific ocean is Tumaco, and traffic in the interior is carried partly on the Patia river (steam navigation to the fluvial port of Barbacoas), and afterwards overland over poor roads by human bearers or on mule-back.

The weight of packages is usually limited to 100 kg., and the freight-cost for heavy pieces is enormous.

The climate is tropical: warm in the lowlands and cold in the highlands; the variation in temperature being about 1° C. for every 160 m. difference of level. The plateau and highlands are very healthy.

Unskilled labor at the mines is principally furnished by Indians, and is first rate; moreover, these Indians and the *mestizos*, once taught, are excellent miners.

The wages are from $0.50 to $1.50 and board (costing $0.50) per man per day, reckoned in silver at about $2.50 per $1 American currency. But both wages and the cost of food vary with the locality, climate, distance from sources of supply, etc.

Vein-mining is of very recent date here. There is no tradition of its existence in Spanish times. On the contrary, the alluvial deposits on the coast (Barbacoas, Mayasquer, Sanabria, etc.) were vigorously exploited, and yielded much gold. These alluvial deposits are very productive just now, being worked unsystematically by scattered parties of negroes.

The predominance of eruptives is the most remarkable feature of the geology of the country; the sedimentaries are, in most places, yet undiscovered. The most common among the eruptives are andesites and basalt-like andesites; these greatly interfering with the metalliferous deposits. Besides the volcanic rocks, igneous rocks of the granite type are very common.

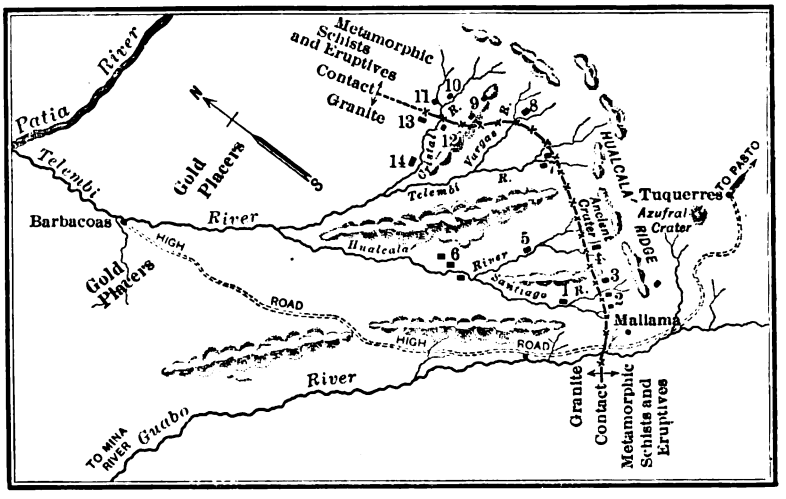
All mining-operations here have gold as an object. Copper-deposits have been found; but no attention is paid to them. Mining in southern Colombia is still in the prospecting stage. Not a single mine is sufficiently worked or developed to furnish thorough information; but the industry is growing rapidly in importance.

II. The Gold-Belt.

The gold-bearing zone has been explored for a length of 50 km. (30 miles) in a general N–S. direction between the districts of Mallama and Samaniago, the southern part being Mallama. It lies on the west side or Pacific slope of the western Cordillera of the Colombian Andes. Fig. 2 is a sketch-map of the Mallama and Samaniago districts.

It is my belief that most of the mines of this belt have had a

common origin, and there is complete evidence that they have been formed by descending waters. In some of the mines the process of formation can be observed. The differences which individualize the mines must be ascribed to the country-rock and to the presence in it of certain minerals (*e.g.*, manganese-ore). The hypothesis of expiring volcanism (of Spurr, if I am not mistaken) receives complete confirmation here.



MINING-CLAIMS.

Mallama District.	Samaniago District.
1. Alaska.	8. Redencion.
2. Porvenir Group.	9. Rocio Group.
3. Alegria.	10. Concordia.
4. S. Pablo.	11. Socorro.
5. Bombona Group.	12. Madrono.
6. Gualcalá Group.	13. Selva.
7. Villas Group.	14. Canillera Group.

FIG. 2.—SKETCH-MAP OF THE MALLAMA AND SAMANIAGO DISTRICTS, COLOMBIA.

The extinct volcanoes, whose craters are still traceable on the ground, are one of the most striking features of the belt. In all probability these volcanoes were active at a period not far from the present. The eruptive waters, carrying the mineralizers, ran upon the surface of the ground, soaking and impregnating the country-rock, and at the same time filling the crevices and joint-planes of the formation. Such is, in synthesis, the most plausible hypothesis to explain the special

character of the majority of the deposits of the belt from Mallama to Samaniago, and other very similar deposits in this country.

In general, the mines of the belt present two types of ore-deposits: 1, the *mantos*, that is, the impregnated rocks transformed into an earthy (saprolitic) material; and 2, the veins, more or less regular in formation.

According to their mineralization, these deposits can be divided into two groups corresponding also, in a certain measure, with their geographical position: (A) The Mallama (southern) group of pyritic ores; and (B) the Samaniago (northern) group of free-milling ores.

In all the veins the gangue is quartz; among the sulphides mispickel predominates, and afterwards common pyrite; then blende, chalcopyrite, and galena in very small quantities, and the telluride of gold rarely. (Some deposits in the granitic rocks characterized by the presence of molybdenum-ores belong certainly to another order of formation, but there are not sufficient data for a detailed description. A superposition of the two is observed in certain localities).

The southern end of the zone is occupied by granites, syenites, and monzonites, one rock passing into the other, but with predominance of the syenitic type. I will call the whole series granites. All these rocks are highly hornblendic, very frequently porphyritic in structure and rich in zircon.

This granitic formation has been covered in great areas by greenstones (augite-andesite, which I have called prophylite in my Spanish writings about this metalliferous zone), ejected from many of the volcanoes now extinct. These basalt-like andesites form many varieties within a common type; and to them I ascribe the origin of the ore-deposits.

In the northern part of the zone clay-schists (lying, as we may conjecture, on the granites) are the country-rock on the top of the hills, passing into slate at the base. Dikes of light-colored andesites, entirely kaolinized, can also be observed.

Complete geological information cannot yet be obtained, since the country is quite unexplored, and covered with woods; but what I have already said may serve to give an idea about the most important points.

Topographically, this belt is the most distorted, broken, and hilly country in the world.

III. The Mines.

More than 2,500 claims have been located in this zone; but very few of the mines are equipped or even explored. The mining-law in Colombia does not compel the miner to perform annually any given amount of work on his claim. To hold the property, it is enough to pay (very moderate) taxes; and, therefore, a great deal of the mineral land is owned by people who do not work the mines, and the underground richness is practically as non-existent.

The Mallama Region.

In this region two groups of mines are worth describing:

1. *The Porvenir Mines.*—These deposits occur at the contact between granites and greenstones, Fig. 3, occupying a very peculiar formation of veinlets in the granite country-rock, and forming a workable mass of ore about 30 m. (100 ft.) wide. The average tenor of gold is $9 American to the ton. All ore-values will be given in United States currency. There are also narrow veins near the contact running into the eruptive; these stringers are highly pyritic and rich. The sulphides are principally mispickel and pyrite intermixed, and associated with small quantities of blende, chalcopyrite, galena, and tellurides.

The pyritic ore frequently assays $238, and the average of all the vein-stuff $47, to the ton. These values are principally in gold (about 30 per cent. being silver).

The Porvenir claims are the highest in the belt, the uppermost workings being 3,400 m. (11,300 ft.) above tide-level. The climate is very cold, and healthy. As the claims cover a large territory and the slope of the Cordillera is very steep, there are within a short distance lowlands with very agreeable climate and well provided with timber and water-power.

The property has been explored a little under my direction, and a small plant (three-stamp mill and concentrator, made by Fried. Krupp) is now under erection. The intention is to cyanide on a very small scale, after roasting the rich pyrites and concentrates.

My laboratory experience in cyaniding these ores was as

follows: After crushing to 60-mesh, I extracted from crude unroasted pulp 80 per cent. of the gold contained in material assaying $9 per ton, and 50 per cent. from $240 stuff. From the latter, after roasting, I extracted 95 per cent.

The free gold in these mines is limited to the outcrops and saprolitic ores; but I think experience with the stamp-mill will prove that there is a great deal of it in the pyrites of secondary enrichment, though no important amount of "colors" can be obtained from them by panning.

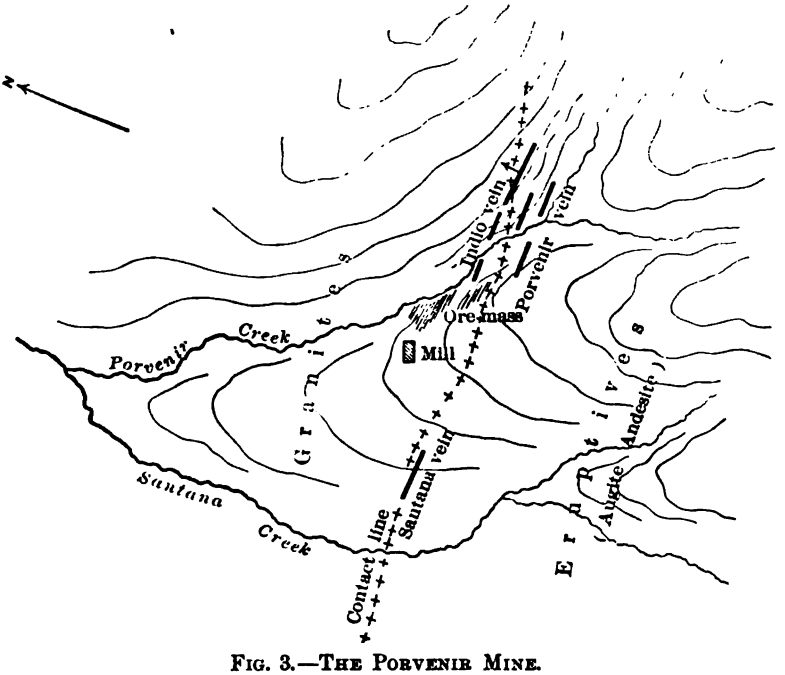


FIG. 3.—THE PORVENIR MINE.

The general direction of the system of parallel veins and veinlets is E–W., and the dip N. from 50° to 60°. The thickness of the veins is very variable, they being lenticular masses, but an average of 0.50 m. (1.5 ft.) can be established, maybe something more. Great irregularities are to be noted, both in formation and in mineralization.

This property is one of the most interesting in the region, and when more fully explored will throw great light on the study of the genesis of the ores. Its formation depends immediately on the extinct volcano of Hualcalá, the crater of which

is within the boundaries of the property. (Following the Inca orthography this name is written with H, but the official name is Gualcalá.)

2. *The Gualcalá Mines.*—Very near the Porvenir group, Fig. 4, and at a lower altitude (2,700 m.), an American company owns a very important group of mines, and has erected an up-to-date all-sliming cyanide-plant. Col. R. A. Linton, an American engineer, is in charge of the enterprise.

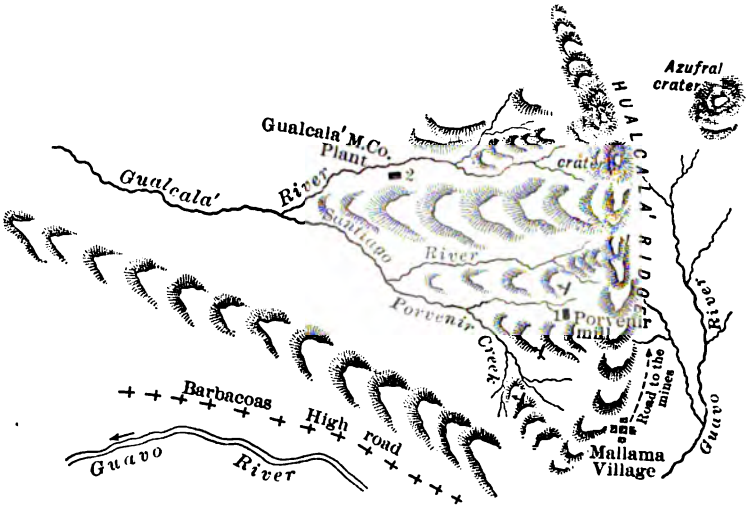


FIG. 4.—SKETCH-MAP SHOWING THE POSITION OF THE PORVENIR AND THE GUALCALÁ MINES.

The property is a system of parallel lodes, some of them very well-defined and regular in formation. The Bombona vein has been most explored, and is somewhat extensively developed in three levels. It is a clearly-defined fissure-vein of an average thickness of 1 m., with a gangue of white quartz and shoots of "black metal"—a complex of mispickel, galena, blende, and pyrites, very rich both in gold and silver. I imagine that the average value of the whole of the vein-stuff may be $30 a ton; in places, very high assays are obtainable.

The wall-rock is monzonite (which has been called also quartz-diorite); at the top of the hill greenstones are found, and it is possible that the same formation as at Porvenir exists there. The Gualcalá Mines Co. has the merit of being the first to introduce the most modern methods of metallurgical treatment into southern Colombia.

The mineral deposits of this part of the belt are subordinated to the extinct volcano of Hualcalá (Gualcalá). This crater, doubtless, was the most important in this part of the country; its ejections covered immense areas, and the crater itself, as it is traceable on the ground, is of enormous dimensions. Some of the remains of its walls form the very picturesque obelisk-like *Pico do Gualcalá.*

The volcanic breccia covers almost all the surface of the territory. This material acted as a filter for the mineralizing waters. It is observable that this breccia is something like "the mother" (*madre*) of the ore-deposits. In the Porvenir mines, for instance, it is easy to see the veinlets formed by the percolation of the materials of the vein through this breccia or *madre.*

The Samaniago Region.

This region attracts attention especially by its free-milling ores, which are historically and economically worthy of interest, as comprising the first worked and the now productive deposits of this part of the Republic.

I shall limit my description to a few of the ore-deposits now operated, especially the Concordia mine, which was the pioneer enterprise. Being now consulting engineer of this property and all its neighbor mines, I am able to furnish trustworthy information concerning them.

1. *The Concordia Mine.*—The country-rock is a clay-schist, altered, on the top of the hills, passing into "writing-slate" at the base of the formation.

The valuable mineral deposits are confined to the upper part; the more saprolitized the schists the better. These clay-schists, in the "good places," are something like natural brick.

The two kinds of deposits already mentioned, *mantos* and veins, are very distinguishable here. The values in the first are somewhat erratic; but the whole mass contains gold from $0.66 per ton to very high grades.

The veins are very peculiar: not sharply defined; very irregular in thickness; sinuous in trend, and generally low in sulphides. The principal lode is a flat vein, reinforced by numberless feeders going from the *mantos* to the vein.

In the rich spots the quartz is smashed as if it had passed through a crusher; and, in general, the gold is loose, not ad-

herent to the gangue—a feature peculiar to these ores, in which it is very rare to find specimens of quartz showing gold, even though extracted from places as rich in free gold as from $600 to $700 per ton.

The *batea* is in this locality the most useful assay-outfit for ascertaining rapidly the quality of the ore, specially for the earthy material of the *mantos*. With some practice, it is very easy, by the aid of this apparatus, to ascertain the values closely.

Probably the veins here belong to a compound system formed by a principal, much distorted, flat vein with feeders and a system of secondary parallel veins. Many of these had been formed before the time of the ore-forming eruption, and not all of them were enriched by that agency. This explains some part of the irregularities observed in the distribution of the values, and the existence of numberless quartz lodes practically barren.

As a rule, the paying veins are narrow, from 0.30 m. to 0.60 m. (1 ft. to 2 ft.) thick; commonly ribboned, but most generally smashed; charged with manganese oxides, and with a tenor in sulphides of from 1 to 2 per cent. The big lumps of solid quartz are of very low grade; the same remark must be applied to the big lodes.

Among the sulphides mispickel predominates, together with the "black metal." The tellurides, though not yet isolated, have been recognized by their reactions and are common in the richer ore. The "headings" from concentration show very variable assays, from $600 to $2,400 per ton. The bullion averages 780 in fineness.

The mine has been badly "picked" in past years by mining only the rich vein-outcrops and the richer part of the *mantos*, regardless of future development. The average value of those ores in the past was from $30 to $50 per ton in free gold. Now the situation is changed, and the mine must be worked and the ore treated by scientific process; but the property still holds its place as a very rich and profitable one.

The metallurgical plant has been the crudest that could be imagined: native wooden mills and silvered copper plates, without any attempt at concentration or provision for further recovery save a very perfect storage of the tailings. At present

a 5-ft. Huntington and a 10-stamp mill, made by Fraser & Chalmers, England, are under erection. The property will probably pass into the control of the American "Gualcalá Mines Co."

The location of the Concordia, Transval and Socorro mines is illustrated in Fig. 5.

2. *The Socorro Mine.*—In general features, this resembles the Concordia; but the deposit has been worked more systematically, principally by open quarrying, with little underground mining.

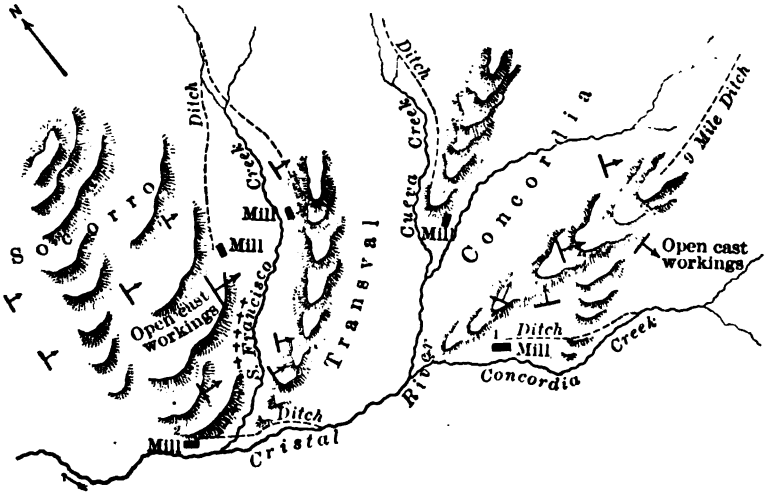


FIG. 5.—SKETCH-MAP SHOWING THE POSITION OF THE CONCORDIA, THE TRANSVAL, AND THE SOCORRO MINES.

The *manto* formation predominates here. The ores are very clayey; manganese is not so common; and no remarkable amount of tellurides is present; and besides these differences from the Concordia mine, there is a greater parallelism and regularity of the general trend of the veins, which is very constantly E–W.

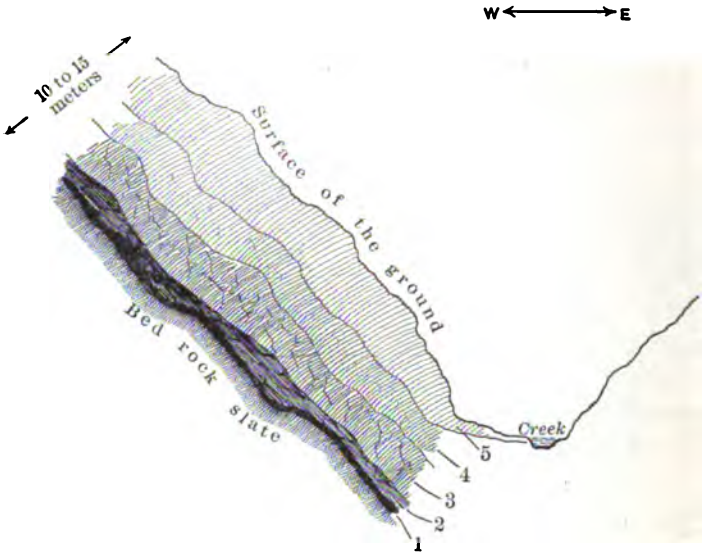
The proportion of concentrates from the milled ore is 0.3 per cent., of an almost constant value of 1 kg. of gold ($667) per ton. This low tenor in concentrates depends on the fact that the greater part of the milled ore is saprolite and shaly material from the *mantos*.

The quartz veins are narrow (maximum thickness, 0.40 m. =

16 in.), beautifully ribboned, and not so crushed as in Concordia. They carry about 4 per cent. in sulphides.

The metallurgical plant is a native mill.

The open-cast workings of this mine are highly interesting for the study of the geology of the ore-deposit, Fig. 6. The cut is about 80 m. high from the level of the creek, and follows the bed of the vein. The schist of the wall-rock is hard and firm, and the dip from 60° to 70° E.



1. Quartz.
2. Ocherous material. Very rich. } *Manto.*
3. Saprolite.
4. Clay-Schist.
5. Soil.

FIG. 6.—GOLD-BEARING FORMATION AT OPEN-CAST WORKINGS IN SOCORRO MINE.

The work is easy and cheap, and the cut goes on the strike, following the bed-rock, with a width of about 15 m. Almost all the stuff pays for milling; the richest part is an ocherous material in the quartz vein. The bed-rock is a little "steppy" in its dip, and on the "steps" are always found rich pockets. As in the Concordia, all the gold is loose in the gangue.

3. *The Transval Mine.*—This mine is located in the ridge between Concordia and Socorro. It is only a prospect, fitted out from the beginning with an Allis-Chalmers five-stamp

mill of the type having 850-lb. stamps. The idea of the promoters of the enterprise was, that if the two adjacent properties, Socorro and Concordia, were so good, the land between them should be also very rich. Experience has not yet corroborated this presumption, and, though well equipped, the mine is not in paying condition.

The system of veins in the Transval is well defined and parallel with the Socorro system; but the ore is very low in free gold, and the pyrites are also poor, which is very remarkable in this locality.

It may be that future explorations under my direction will prove the existence of rich ore like that of the neighboring mines; but for the present only very sound metallurgy can make things right.

4. *The Madrono Mine.*—Down in the direction of Rio Cristal, the slate formation gives place to the granites at about 5 km. (3 miles) from Concordia. In this country-rock many mines have been worked, with not very favorable results. The most fully explored of them is the Madrono.

The deposit is more like a stock-work than a system of veins. The granites are altered to a great depth, not only by weathering, but also by dynamic action, and are crossed by veins and veinlets which in places reunite in knots, forming great bunches of ore.

The quartz is sugary in texture and the gold is adherent to the gangue. The distribution of the pay-shoots is very erratic; as in all the gold-formations here, the country-rock contains values—generally very small, but, in the vicinity of rich pockets, in paying amount.

Beautiful specimens of native gold (foliations, plates, dendrites, etc.) have been found occasionally in this mine.

With the exception of the Transval, all the mines are provided with native mills, and the ores are handled and milled in the crudest fashion. All transportation has been performed heretofore on men's backs, etc. I have now erected some cable-lines at Concordia and rail-lines in Socorro and Transval, which entirely change the transportation feature in the economy of the undertakings; but the most important reforms in method remain to be effected, viz.:

1. The transportation of the *mantos* (earthy ores) by means of water in sluices or, better, in pipes.

2. Preliminary washing in trommels (fine screening) of the ores and immediate recovery of the fine loose gold on plates.

3. Hydraulic exploitation of a great part of the deposits (*mantos*).

4. Concentration of the trommel- and mill-tailings and final recovery of the concentrates-values by cyaniding or by exportation of the high-grade concentrates.

IV. Conclusions.

A few observations are suggested by the foregoing statements.

(*a*) As a rule, the big quartz lodes are almost barren (say from $2 to $5 per ton) in free gold.

(*b*) Very frequently the big veins are accompanied by feeders (called here *aqujas*), which are generally rich.

(*c*) The richest parts of these deposits are the narrow, not well-formed, veinlets, much like lenticular masses.

Farther NE. of the group of mines I have hastily described, new prospects are attracting considerable attention; small native mills have been erected; and the exploitation of rich *mantos* is giving fair results.

The discovery of permanent gold-bearing formations, the richness of the outcrops, and principally the *mantos* with their extraordinary facilities for cheap working, warrant the expectation that a very important future is reserved for these metalliferous zones of southern Colombia.

Contribution to the Study of the Pre-Cambrian Rocks of the Harney Peak District of South Dakota.

BY GORDON S. DUNCAN, A.R.S.M., A.I.M.M., LONDON, ENGLAND.

(New York Meeting, February, 1912.)

THE U. S. Geological Survey, I believe, has almost completed a study of the Harney Peak quadrangle, preliminary to the publication of a report on that district. As I was engaged for some months on an investigation of the mode of occurrence of the ore-bearing formation in this area, it has seemed to me timely to make public the results of my study, as a contribution to the consideration of this subject.

It seems scarcely necessary to describe the locality and general geology of the Black hills of South Dakota, for since 1846 many have contributed information concerning this region, some of the reports bearing such well-known names as those of Walter P. Jenney, Henry Newton, S. F. Emmons, and C. R. Van Hise, while we have only to glance at *Bulletin No. 4 of the South Dakota School of Mines*, in which Prof. C. C. O'Harra, of that institution, has given us a bibliography of contributions to the geography and geology of the section, to find that other well-known men have studied the district, and have given us the benefit of their researches.

The U. S. Geological Survey has made geological maps of nearly the whole of the Black hills, the only quadrangle yet to be completed being, as stated, the Harney Peak district. During the late summer and autumn of 1910, it was my lot to examine and report on a considerable area in this neighborhood, which once had been prospected, and in some cases exploited, for tin. This paper gives an account of certain geological phenomena, noted at that time, which greatly influenced my conclusions.

The predominating formation of the district is pre-Cambrian, represented by a highly-cleaved mica-schist, usually very garnetiferous, and sometimes hornblendic. Rising like an island from the surrounding schist is a large mass of granite, known as

Harney's Peak. Originally the whole of this area was overlain by the Potsdam conglomerate, Carboniferous white limestone, and representatives of the Jurassic and Cretaceous series, the escarpments of which are prominent features in the landscape surrounding the pre-Cambrian formation, and which must have represented many thousands of feet above the present horizon. Erosion has therefore been enormous where the pre-Cambrian rocks outcrop. As well as the main occurrence of granite, there are numerous outcrops of pegmatite, greisen, and quartz throughout the schist formation. These appear as veins or ledges, in some cases parallel with the cleavage of the schists in their strike, in other cases cutting across it, and thereby causing some observers to consider them to be true fissure-veins. It was to these ledges that my attention was principally directed during my investigation, since it was here that cassiterite had been found.

In this district, cassiterite occurs in a rock containing usually two of the constituent minerals of granite (*i.e.*, in a quartz-mica aggregate known as "greisen"), and more rarely in quartz alone. The greisen itself occurs in two forms:

(1) *Massive*, occupying the whole of the vein, the constituents being themselves arranged in a markedly segregated manner, as illustrated in Fig. 1, which shows an ideal section of a greisen ledge. Here cassiterite occurs on either of the walls, in association with the mica, or else in the center, where the largest crystals of mica usually are formed. It is rare to find cassiterite in the portions of the vein composed almost entirely of quartz, although occasionally large crystals of the mineral are met with there. It is, however, a fairly general rule, that where large crystals of cassiterite are found, the mica is in large crystals also, which at least suggests that the presence of cassiterite is not due to subsequent deposition.

(2) *Granular.*—This form of greisen is a fairly homogeneous mixture of quartz and mica, and is associated with pegmatite, occurring in ledges of that rock in irregular pockets and shoots, usually few and far between, and generally at the contact of the pegmatite with the schist. It seems to contain the more cassiterite the more mica is present, and always in smaller crystals than in the massive greisen.

Fig. 2, a section of one of the ledges, is a good example of

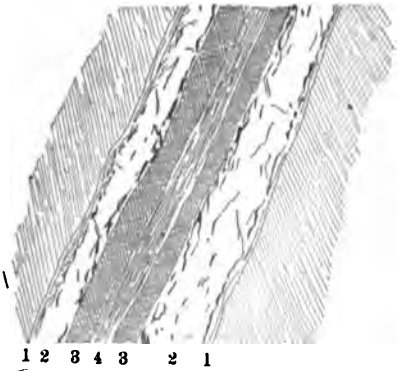
the manner in which granular greisen occurs in a pegmatite ledge.

I would note here that the presence of greisen, massive or granular, does not always indicate the presence of cassiterite, since I never noticed cassiterite in any rock where the three constituent minerals of granite were present or where disseminated tourmaline occurred in any quantity.

There are three types of ledges in this district:

(1) Those consisting of pegmatite containing erratic occurrences of granular greisen, usually from 3 to 20 ft. wide.

(2) Those consisting of massive greisen only.



1. Small crystals of mica near contact, and some cassiterite.
2. Quartz and a little mica disseminated through it in cracks.
3. Small mica crystals, becoming larger towards the center of the vein; very little quartz.
4. Much mica, very little quartz, and most of the cassiterite.

FIG. 1.—COWBOY LODE.

(3) Those consisting of quartz only, carrying little or no mica, practically no tin, and generally only a trace of gold, although in some cases gold in paying quantities has been discovered in large pockets or "bonanzas," distributed very irregularly throughout the ledge, as is usually the case with the white glassy quartz such as occurs in this region.

The relations between these ledges and the country schists and the main granite intrusion are the two principal points to which I wish to draw attention.

The difference in texture, and to some extent in composition, between the granite batholith of Harney Peak and the ledges

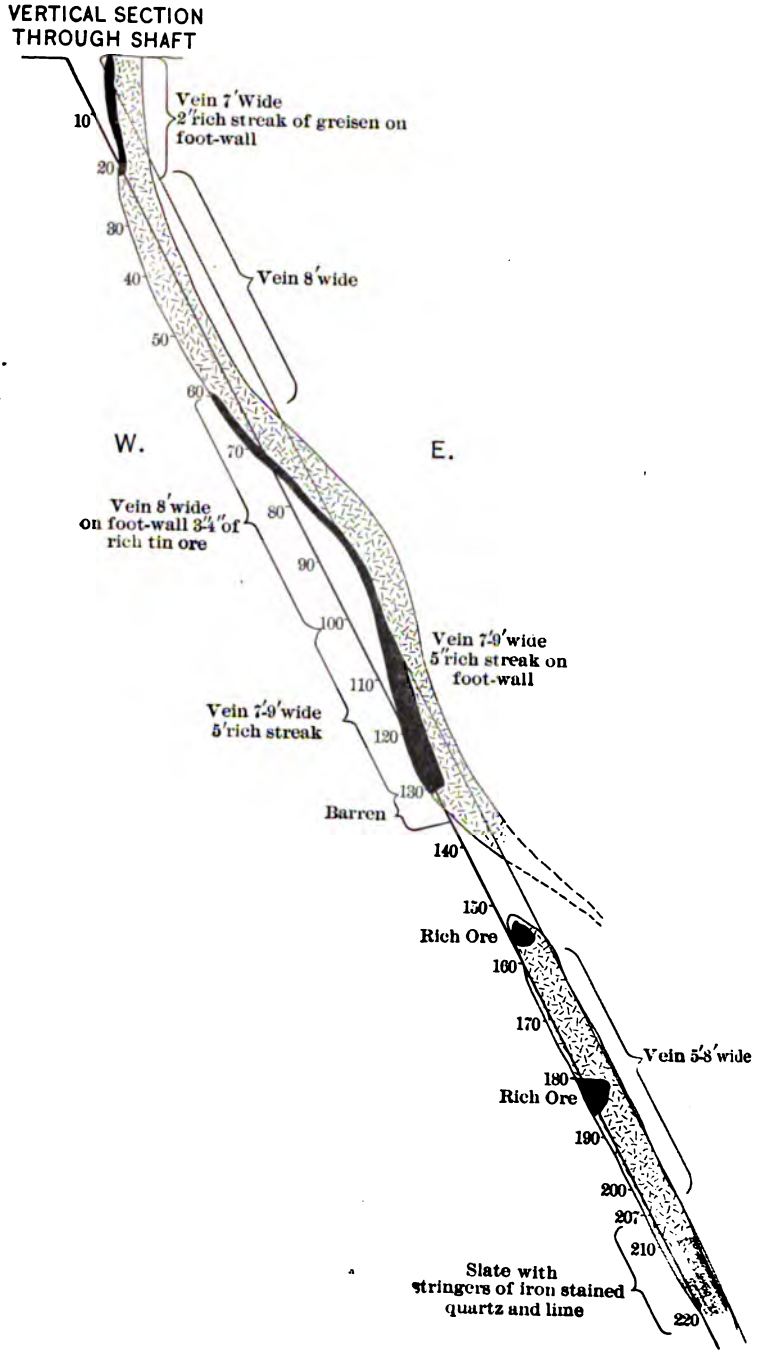


FIG. 2.—ADDIE MINE. (After T. Ulke.)

occurring in the vicinity is very marked. The granite of the former is for the most part of the coarse-grained normal variety, having the feldspar in white crystals, somewhat larger than those of the quartz and mica, and carrying much tourmaline of the usual deep blue variety. Never did I find in the granite any greisen, or any trace of cassiterite. Rarely does one see on Harney Peak any evidence of segregation of the minerals of the granite, beyond the fact that the feldspar is usually inclined to be idiomorphic and to occur in somewhat large crystals. The arrangement of the crystals shows in places a tendency to gneissic structure. Passing away from the central granite batholith, we find outcrops of many granitic dikes, which show a somewhat marked structure in the arrangement of the crystals of the constituent minerals. The feldspars are all crystallized out in large idiomorphs; much tourmaline and a little columbite are present. Proceeding further westward, we pass outcrops of granite lenses, becoming very coarse-grained and showing a distinct foliation in their structure, almost resembling a pegmatite. Many outcropping lenses of quartz can be noticed. The mica-schist in this vicinity dips very steeply westward and masses of it stand out in high relief from the surrounding country, especially where they contain a core, as it were, formed by an included lens of quartz or pegmatitic granite. Continuing in a westerly direction, we find that the granite ledges begin to show marked segregation of their constituent minerals, and to present more of the texture of pegmatite. The quartz is inclined to be more segregated than its associates and to occur in veins and masses in the pegmatite, while now and again granular greisen is met with, sometimes showing crystals of cassiterite. The greisen, however, is very local in its occurrence, here as elsewhere in the district, where it occurs in a pegmatite. A minute examination of one of these ledges would reveal a length of only a few feet of greisen (usually at the contact with the schist walls) in several hundred feet of the outcrop. As we proceed further westward, and away from the main granite intrusion, the quartz veins are more frequent, both in the schist and in the pegmatites, and at a distance of about 4 miles from Harney Peak veins or lenses of massive greisen occur. We see, therefore, from the foregoing, that the further away from Harney Peak,

the more segregated, generally speaking, are the minerals, especially the quartz, in the pegmatites.

Southwest of Harney Peak, and in the Custer City district, we find a greater tendency than to the N. and NW. of the granite intrusion towards a more equal segregation of all the minerals of the pegmatites. The result is that large masses of quartz are less frequent in this district than they are at an equal distance from the granite intrusion at Hill City, although even here the quartz is inclined to segregate itself to some extent in the pegmatites. For example, the New York mica-mine, owned and operated by the Westinghouse Electric & Manufacturing Co., has opened up a lens of pegmatite 30 ft. wide, which is distinctly more quartzose at the center than at the contact with the schist; but the largest crystals of mica are to be found within 12 ft. of one schist wall and within 4 ft. of the other. This is not always the case; for at another of this company's mines the pegmatite is productive of large sheets of mica, usually from wall to wall. Comparatively few of the many pegmatite lenses in this district have crystallized coarsely enough to make the mica contained in them marketable. By far the greater number are of much finer texture. But there seems in this southern district to be no very marked gradual change from fine to coarser crystallization as the distance from the granite increases. The extremely coarse-grained pegmatites are dotted very sparsely among others of more normal texture; and, in fact, some are in close proximity to the granite itself.

A feature common to almost all the bodies of pegmatite, and also to the schist country in which they occur, is that the strike of the former and the cleavage of the latter are in most cases roughly at right angles to a line drawn from the center of the granite intrusion of Harney Peak. This remark is not intended to imply that the strike of the cleavage of the schists and that of the pegmatites, greisen, and quartz ledges is in every case coincident. There is sometimes a difference of from 10° to 40° between the two. Moreover, the schists almost always dip at a high angle away from the granite, while the ledges may dip either one way or the other; and in many cases where the dip of both ledge and schist is in the same direction, the angle is very different. The fact that

the schists always tend to dip away from the granite and that their strike (except to the north of Hill City) is rudely parallel to the sides of the granite intrusion, indicates that the metamorphism of the schists has been to a great extent due to the intrusion of this mass of granite. But that the whole of the metamorphism of the district has been caused by this one factor is most improbable.

The ledges themselves furnish some evidence in this connection. I take as an example the ledge of massive greisen which has been developed in the Cowboy mine, near Hill City.

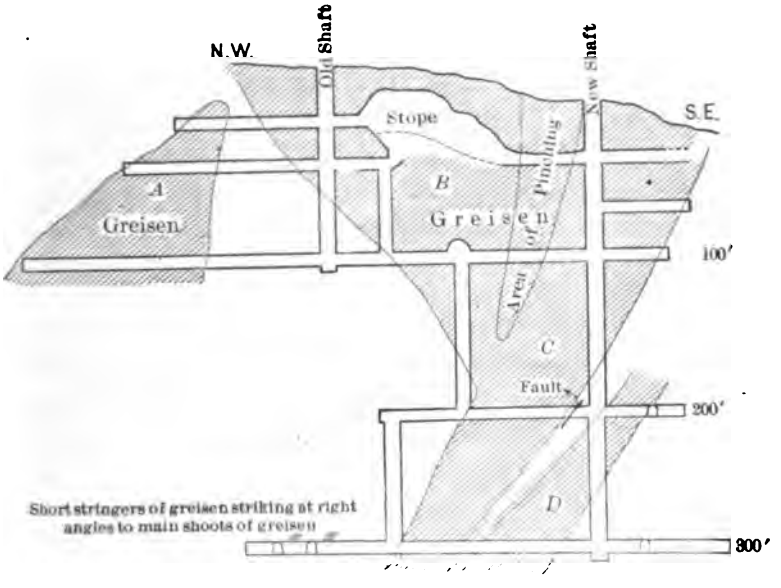


FIG. 3.—SECTION OF COWBOY MINE.

The arrangement of the mica and quartz is shown in Fig. 1. There is a difference between the strike of the greisen and that of the schist of about 30°. The former dips about 75° NE., while the latter dips about 65° NW. This might suggest a fissure-vein, were it not for the fact that the greisen has the appearance of being "frozen on" to the schist walls.

The longitudinal section of the mine, Fig. 3, shows the various shoots of greisen, and it will be seen how very lenticular they are in depth. The schist walls show numerous slickensides in many directions, and in some places slight contor-

tions in dip and strike. A distinct fault is noticeable on the 200-ft. level, where the greisen of shoot *C* has been turned round at right angles near the shaft, and has "petered out" in the west wall, while on the 300-ft. level, at the far north end, two little stringers of greisen, with a strike at right angles (roughly) to that of the main shoots, have been followed by tunnels, which proved only a rapid "petering out" in each case. This clearly shows that a force acting at right angles to that which had produced the cleavage of the schists, had been applied to the rocks in the vicinity, after the deposition (if deposition we may call it at this stage) of the greisen.

A neighboring shoot of greisen, which strikes at an angle with the schists, shows the effects of this subsequent dynamic action, being crossed from wall to wall by cracks following the line of strike of the schists. Besides these two instances in which a second lateral disturbance is evidenced in the greisens, there are many others to be seen in the pegmatites also, although here we find the cracks in the ledges running rather at right angles to their strike than in the direction of the cleavage of the schists.

As I have already observed, the pegmatites showing greisen usually do so on the walls—*i. e.*, at the contact of the granitic rock with the schist. It is also not unusual to find the greisen in a crack or fissure in the pegmatite, in which case it is generally massive and rarely granular.

Possible Causes of Dynamic Action Subsequent to the Intrusion of the Main Granite Batholith.

As we have already seen, there is ample evidence to prove that, besides the intrusion of the Harney Peak granite, there were forces, acting in this region at a subsequent time and in other directions, which cracked and fissured the ledges of quartz, pegmatite, and greisen.

What produced these forces?

North of Harney Peak, and within a mile or so of the granite itself, we find several outcropping bosses of pegmatite very much coarser in texture than any of the ledges hereinbefore described. Their appearance suggests also that, unlike those ledges (which have been proved by underground prospecting in several places to have no greater continuity in

depth than on the surface), they may have considerable downward extension, and even a deep-seated origin. I refer to masses of the type of the fairly well-known Etta boss or knob, which was the first to receive attention during the tin-excitement in the late nineties. Here we have a pegmatite (giving evidence which points to an intrusive origin), the constituent minerals of which have crystallized out on an enormous scale.

Professor Blake, who reported on this property about 1883, described the arrangement of the minerals in the boss as showing a tendency to more or less concentric segregation. This is to a certain extent true, although the lines of demarcation are less hard and fast than Blake's report suggests. He describes the zones as follows: (1) a central core of quartz and feldspar; (2) greisen, carrying tin-stone; (3) crystals of spodumene, with greisen, carrying tin-stone; (4) a band or zone of biotite and muscovite mica.

Spodumene is found, in this region, only in bosses of pegmatite of extraordinary coarseness, like this one. Furthermore, this intrusion contains not less than 48 distinct minerals, many of which are absent from the more common and more normally crystalline pegmatite ledges. Of the minerals occurring here, the most important, as regards size of segregated masses, are quartz, albite, spodumene, and mica (the latter being usually commercially useless, since it occurs as muscovite and biotite in "fishbacked" plates). Granular greisen is also prevalent, and in some spots, especially near the center of the boss, exhibits a certain amount of cassiterite, while aggregates of small crystals of mica occur in association with it.

That such occurrences of pegmatite have no connection with the main intrusion of granite, there is scarcely sufficient evidence to prove, but it does seem possible that their intrusion was connected with the action which produced some of the later lateral disturbances and uplifts. They have no real parallel southwest and south of Harney Peak, where the very coarsely crystalline pegmatites productive of commercial mica outcrop at various distances from the granite. There are some marked differences between the two groups. The latter show relatively fewer minerals; often occur as lenses in the schist, like the more normal pegmatites; and the mica in many of them is the variety known as phlogopite, which is generally

common to most of the ledges in the near vicinity, and is the mica of some of their tin-bearing granular greisen. There seems to be, therefore, a distinction between the mormal pegmatites occurring in the Harney Peak district all round the granite, and the highly segregated intrusions to the north; and the question suggests itself, in view of the evidence of subsequent dynamic action portrayed in the fissured normal pegmatites and greisens, whether these other intrusions of coarser granitic material outcropping to the north are not the cause, or products of the cause, of the later disturbances and uplifts thus shown to have taken place. In connection with this question, the schists and slates which constitute the country-rock of the district may be considered.

Newton, in his report on the geology of the district, seems to think that the present surface-exposure of schists and slates represents from east to west the original thickness of the pre-Cambrian series. He says, in speaking of the body of Archæan slates and schists:[1]

"It has not, of course, its original position; for the same process of change which has produced its metamorphic structure has turned it bodily on edge, and either broken or eroded away its upward continuation; . . . and it is probable that the system presents the clays, shales, and sandstones from which it was produced by metamorphism in the same order in which they were originally deposited."

Apparently, Newton thought there was some connection between the cleavage and foliation of the schists and the original bedding of the detritus from which they were formed. Against this supposition we have the following important facts:

1. The general strike of the cleavage of the slates and schists is parallel to the main limits of the granite batholith.

2. The general dip of the slates and schists is away from the granite.

3. The dip of the schists is generally extremely uniform. Had the present structure represented bedding, this state of things would have been most improbable; the folds would have had to be very closely pressed, and, as Van Hise says,[2]

[1] The Pre-Cambrian Rocks of the Black Hills, by C. R. Van Hise, *Bulletin of the Geological Society of America*, vol. i., pp. 203 to 244 (1889).

[2] *Idem.*

"the sides . . . would have had to be truncated in such a fortunate position as to cut none of the folds at their turning-points."

4. In only one instance has a true anticlinal fold of large size been noted in the schists. This is in the neighborhood of the very coarsely crystalline pegmatites in the Custer City district, and is probably due to a purely local disturbance.

We can therefore say that the available evidence all points to the fact that the present structure exhibited in the pre-Cambrian slates and schists is the product of metamorphism, and has no relation whatever to their bedding; and, furthermore, that the forces producing the metamorphic structure, which have acted at various times and in various directions, have completely obliterated all traces of the original bedding of the detritus. Many of the pegmatite occurrences in the schist exhibit a marked flow-structure in the arrangement of their minerals. This is probably due to forces which have acted on them subsequent to their deposition. None of the ledges of pegmatite, quartz, or greisen have any great continuity along their outcrops. In some cases, where shafts have been sunk on them to a considerable depth, this has been found to be true as to their downward extension also. Most of those upon which the greatest development-work has been done show somewhat irregular pinching and widening and faulting, although the schist itself may not exhibit any marked contortion in the immediate vicinity—the only trace of the course of a fault underground being, in such cases, a very narrow gouge or clayey stringer.

Origin of Lenses of Pegmatite and Quartz.

The study of the origin of the pegmatite, quartz, and greisen in this district raises the question of the probable causes of pegmatization. Van Hise, in his work,[3] states three causes for this phenomenon, all of which may be applicable to the Harney Peak district, namely: (1) true igneous injection; (2) aqueo-igneous action (intermediate between solution and fusion); and (3) water-cementation.

As to the first cause, we have, in close proximity to the main granite intrusion of Harney Peak, several very large dikes, of composition and texture extremely similar to that of the gran-

[3] A Treatise on Metamorphism, *Monograph XLVII., U. S. Geological Survey*, pp. 720 to 728 (1905).

ite batholith, but exhibiting a distinct gneissic structure. It seems impossible not to imagine that these dikes are connected in depth with the main granite intrusion, and owe their origin to igneous injection.

Further away from the main granite, the minerals of the pegmatites become more and more segregated, and there is a distinct gradation from finer to coarser texture (although with very little degree of regularity in the ratio of increase in distance). Some of these occurrences of pegmatite, usually those in which a small percentage of cassiterite has been found, have been prospected for that mineral by shafts, which, in several cases, have proved their lenticular character in depth as well as in length by "bottoming" them. To these it seems reasonable to ascribe an aqueo-igneous origin, through the injection, along fissures, in a state between fusion and solution, of material more acid in composition than the granite magma, which has crystallized very coarsely when deposited. Van Hise puts forward the theory that the coarseness of crystallization at a distance from the main granite intrusion is not necessarily due to any great slowness in cooling, but rather to the aid to crystallization afforded by the abundance of mineralizers present.[4] This theory certainly answers for the majority of the phenomena of pegmatite lenses in this immediate district. Still further west and north from Harney Peak, the lenses of pegmatite become more quartzose and the number of lenses of pure quartz increases. West of Hill City we have several lenses of massive greisen or quartz containing mica, both materials being arranged in layers parallel to the walls of schist. The origin of these seems to be accounted for more easily by water-cementation than by any other cause; they are very lenticular and irregular in their occurrence, both in strike and in depth.

It is evident that no hard and fast line can be drawn where the effects of any one cause of the formation of the lenses of quartz and pegmatite begin or end. Their appearance seems to present a gradual gradation from the granitic dikes presenting a gneissic structure, owing their origin to igneous injection, and which occur in close proximity to the main granite intrusion, to the lenses of pegmatite, and thence to those lenses composed almost, if not entirely, of quartz, which occur remote from the granite, and probably owe their origin to water-cementation.

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[4] *Monograph XLVII., U. S. Geological Survey*, p. 722 (1905).

The Occurrence of Silver-, Copper-, and Lead-Ores at the Veta Rica Mine, Sierra Mojada, Coahuila, Mexico.

BY FRANK R. VAN HORN, CLEVELAND, OHIO.

(New York Meeting, February, 1912.)

I. Introduction.

In the summer of 1908, R. B. Cochran, Superintendent of the Compania Metalurgica Mexicana at Sierra Mojada, Mexico, presented to the Department of Geology and Mineralogy at Case School of Applied Science an interesting series of silver-, copper-, and lead-ores from the Veta Rica mine. These ores contained several interesting minerals which were associated in a somewhat peculiar manner, so that I was impelled to make a closer study of their occurrence. The investigation has brought out several new facts concerning the geology of the district and the geological occurrence of the ores, and therefore has seemed worthy of record. Two papers have been published previously about the region, one by Chism[1] in 1887, and the other by Malcolmson[2] in 1902. These articles, however, treat of the district more from the stand-point of the metallurgist and mining engineer, and not much attention is devoted to its mineralogy and economic geology.

II. Geographical Location of the District.

The Sierra Mojada is situated in the extreme western part of the State of Coahuila, near the Chihuahua boundary. It is reached best via the Mexican Central railroad from El Paso, Tex., to Escalon, Chihuahua, a distance of 494 miles. From this point, a branch line, the Mexican Northern railroad, runs 78 miles in a NE. direction to the mines, which are its northern terminus. The railroad was built about 1890, practically on account of the Sierra Mojada mining-district.

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[1] Sierra Mojada, Mexico, by Richard E. Chism, *Trans.*, xv., 542 (1886-87).

[2] The Sierra Mojada, Coahuila, Mexico, and Its Ore-Deposits, by James W. Malcolmson, *Trans.*, xxxii., 100 (1901).

III. Climate and Topography.

The region is an arid one, and therefore the supply of water is small. The rainy period may occur between the middle of May and the middle of July, but during some seasons there is practically no rain. Two springs are known, and productive wells may be obtained by deep borings; but it has been found more economical and satisfactory to ship water in from points beyond Escalon on the Mexican Central railroad. The timber for mining purposes is also obtained elsewhere. On account of the conditions just mentioned, the region surrounding Sierra Mojada is an uninhabited desert.

Sierra Mojada is the name of a town as well as of a range of mountains or hills. The town lies in a valley from 2 to 3 miles wide N–S. and about 12 miles long in an E–W. direction. The altitude is about 4,800 ft., and the valley is bounded on the north by hills known as the Sierra Planchada. The southern boundary consists of the Sierra Mojada mountains, which rise more than 2,000 ft. above the valley. These mountains, which possibly more properly should be called hills, rise gradually out of the valley with the exception of the last third of their height, which ends in a steep cliff of about 600 ft. This precipice is found for a distance of 3 miles in the highest portion of the Sierra, and dies down into rounded hills on the east and west.

IV. Local Geology.

The rocks which outcrop in the region are limestone and conglomerates, while underground workings show the presence of a so-called breccia.

1. *Limestone.*—The rocks which form the Sierra and constitute the chief underlying mass of the district are somewhat dense grayish-blue magnesian limestones of Cretaceous age. Thin sections show a general similarity to the Palæozoic limestones of the central and eastern United States. They are often stained yellow and red by iron oxide. According to Chism, these rocks dip about 35° S., but in places they stand nearly vertical.

2. *Conglomerate.*—In the bottom of the valley, a limestone conglomerate is found which is clearly the result of the disintegration of the limestone cliffs of the Sierra. The cement of

the conglomerate is chiefly calcium carbonate. In the Veta Rica central shaft, the rock has a thickness of 30 m. It is still locally called "wash," and was called "alluvium" by Chism; but Malcolmson says that in at least three places this conglomerate has resting upon it a still younger Cretaceous limestone, which, in one locality, on a *mesa* known as La Blanca, is 100 ft. thick. This proves that the valley was formed in Cretaceous time, and depressed beneath the sea during the same period. The first discovery of silver-lead ore in 1878 was made, and the first claim was located, in the conglomerate on La Blanca *mesa*. This seems to indicate that the ores were deposited in the older limestone prior to the submergence and deposition of the younger limestone. These two periods of deposition with the intervening long erosion-period seem to correspond to the great unconformity between Lower Cretaceous (Comanche) and Upper Cretaceous in the Southwest.

3. *Breccia.*—Mine-workings show that, between the calcareous conglomerate and the older limestone, there is a reddish-gray granular rock which is called locally conglomerate, but which Malcolmson calls a porphyritic breccia. This rock has a thickness of 106 m. in the Veta Rica central shaft. I had at my disposal only a drill-core about 5 in. long, a thin section from which showed a somewhat clastic appearance. There is a great deal of granular quartz present, which is in places slightly rounded, and has at times been added to by a deposition of quartz having the same extinction as the original grain. Some of the quartz is also angular, and shows the conchoidal fracture. Other grains consist of badly-decomposed feldspar (apparently orthoclase) which is filled with dark spots, giving a micropoikilitic appearance, probably due to later impregnation with iron oxides. Several Carlsbad twins were observed. Still other grains resemble the microgranitic ground-mass of volcanic rocks. A few grains of a mineral resembling magnetite and some decomposed ferro-magnesian mineral are present in small amounts, as is also calcite. A faint reddish stain of iron oxide is present in most of the grains with the exception of quartz. The rock had two semi-angular inclusions within the limits of the drill-core. One was reddish brown, and the other was purplish red. Both were very fine-grained and massive, resembling volcanic igneous rocks. I am unable to say how

frequent these inclusions are, but if there were many, the rock as a whole would have a distinctly brecciated appearance. To my mind, the chief rock which makes up the drill-core is decidedly similar to a rhyolite tuff, and I am not absolutely certain that it could not be a very much decomposed rhyolite with inclusions. The rock is too badly decomposed, and other data are insufficient, for accurate determination; but it is deserving of further study. This rock is fairly impermeable, and wells are sunk into it through the conglomerate in order to obtain water. It rarely, if ever, outcrops at the surface, and its contact with the underlying limestone has generally a steep dip, although in places it becomes nearly horizontal. Slickensides are found near the contact in both breccia and limestone, and the limestone below the contact is considerably broken. The plane of the contact dips northward away from the mountains, and the ores are found either on the contact between limestone and breccia or in the limestone below the contact.

The formation of the present valley, as explained by Chism, is due to the erosion of an anticlinal fold. His principal reason for this conclusion seems to be, that the limestones of the Sierra Mojada dip southward 35°, while the rocks of the Sierra Panchada have a northward dip of from 25° to 30°. The underground workings show nothing of this anticline, and, at the time of Chism's visit, the breccia was not recognized. Malcolmson accepts Chism's explanation, but observes that the folding was not so simple, because in places, notably above the San José mine, the limestones were bent nearly vertical, or almost at right angles to their normal dip. He says, "it is evident that faulting and dislocation of the strata have also taken place to some extent," and contents himself with that statement, although otherwise he agrees with Chism. Emmons,[3] in a discussion of Malcolmson's paper, suggests that similar structures might also be produced by faulting. My observations all point towards the latter conclusion, and it seems more probable that the valley was formed by faulting, and that the breccia is part of a fault-block originally formed at the surface some distance away. The breccia was later covered by the conglomerate and younger Cretaceous limestone. Evi-

[3] Discussion of Paper of James W. Malcolmson, by S. F. Emmons, *Trans.*, xxxii., 566 (1901).

dences pointing to this origin are the slickensides in both breccia and limestone, and the fact that the limestone for some distance from the contact was considerably fractured and later was impregnated with ore. It is also thought that the limestone cliffs represent the fault scarp which has suffered but little erosion since the dislocation. The plane of contact between breccia and limestone is generally steep, although in some places it becomes quite flat, and also at times is curved, or "rolls," which is not true of fault-planes in general. This, in my opinion, is the greatest objection to the theory of faulting. Some variation in dip might be accounted for by cross-faulting, which has been found at least once on the Veta Rica property, with a throw of 40 feet.

There is a third theory which is not absolutely impossible. Attention has been called to the fact that the breccia has considerable resemblance to a decomposed rhyolite or rhyolite tuff. Either of these might have come to rest upon an eroded limestone surface, which might account for a varying pitch of the contact-plane. In this case, the valley would have been formed by erosion, but not of an anticline. Not enough evidence is known by any one to warrant a satisfactory conclusion concerning the origin of the breccia, and, in my opinion, an accurate investigation of the rock in place is needed to throw light on its true character. This in turn will do much towards explaining the origin of the valley, as well as that of the ores.

Borings on the Vulcano and El Acaso claims, north of the Veta Rica, show that a rock called locally "sandstone" or "quartzite" is encountered below the limestone at a depth of 327 and 346 m. respectively on the two claims. Underneath this rock, a "slate" is said to be found at 440 and 455 m. These rocks show a slight dip to the north rather than to the south, and seem to have no relation to any anticlinal fold.

V. Position of Mines and Character of Ores.

The mines of the district are situated in the foothills about one-third of the distance from the floor of the valley to the summit of the Sierra. There are from 15 to 20 in number, distributed approximately on an E–W. line for about 3 miles. The three largest producers in 1900, according to Malcolmson,

were: Veta Rica, 41,018; San José, 30,285; and San Salvador, 28,783 metric tons.

The general character of ores and their composition were given by the same author as follows:

	Metric Tons.	Ag. Oz. Per Ton.	Pb. Per Cent.	SiO_2. Per Cent.	FeO. Per Cent.	CaO. Per Cent.	$BaSO_4$. Per Cent.	Cu. Per Cent.
Basic lead-ore,	80,000	9.0	15.0	16.0	36.0	5.0	—	—
Siliceous lime,	52,000	18.0	0.0	20.0	3.0	21.0	10.0	0.6–3.0
Lime ore,	25,000	8.0	0.0	7.0	3.0	33.0	—	—
Siliceous lead-ore,	23,000	10.0	14.0	34.0	22.0	2.0	—	—

The chief ore of the district has always been a silver-lead ore consisting largely of argentiferous cerussite, but since 1893 the mines in the western part of the camp, notable among which are the Veta Rica and San José, have produced large amounts of silver-copper ores, containing little or no lead.

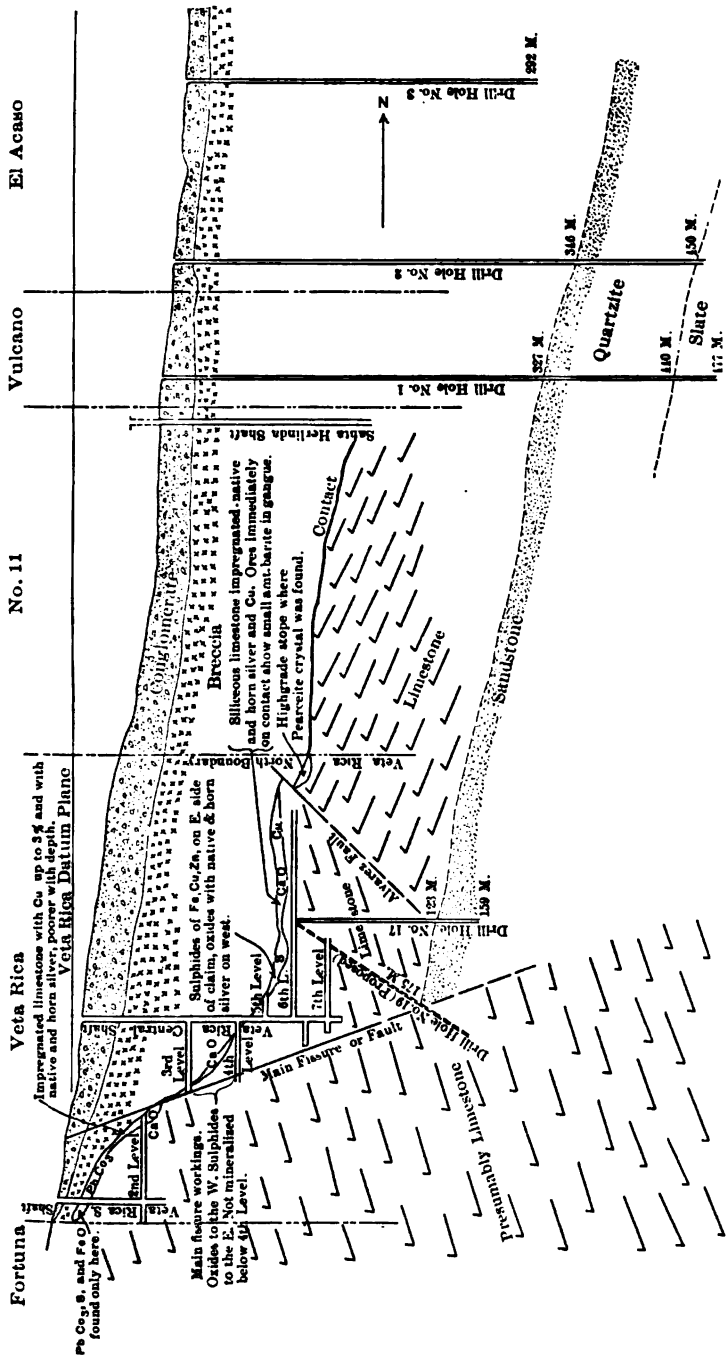
VI. Veta Rica Ore-Bodies.

The Veta Rica claim covers 9 hectares, or about 22.25 acres, and considerably more than $8,000,000 has been taken from the mine to date. The first shaft was sunk near the southern boundary with the adjoining Fortuna property, and was carried only to the 2d level (see Figs. 1 and 2). At the 1st level, the ore common to the camp as a whole was encountered. This was argentiferous cerussite mixed more or less with limonite, massive sulphur, and gypsum. This association is quite usual in the region, and the sulphur has been obtained in such large amounts that some shipments have been made. The gypsum especially was so constant that the miners used it as an indication of proximity to good ore. The original mineral was undoubtedly galena, and the reaction which took place was probably the following:

$$PbS + 4O = PbSO_4$$
$$PbSO_4 + CaCO_3 + 2H_2O = PbCO_3 + CaSO_4 . 2H_2O$$

There is, of course, a possibility that anhydrite may have been the original mineral, which has altered to gypsum as described by Lindgren.[4] In the present instance, since nothing but secondary minerals are found, the gypsum was probably di-

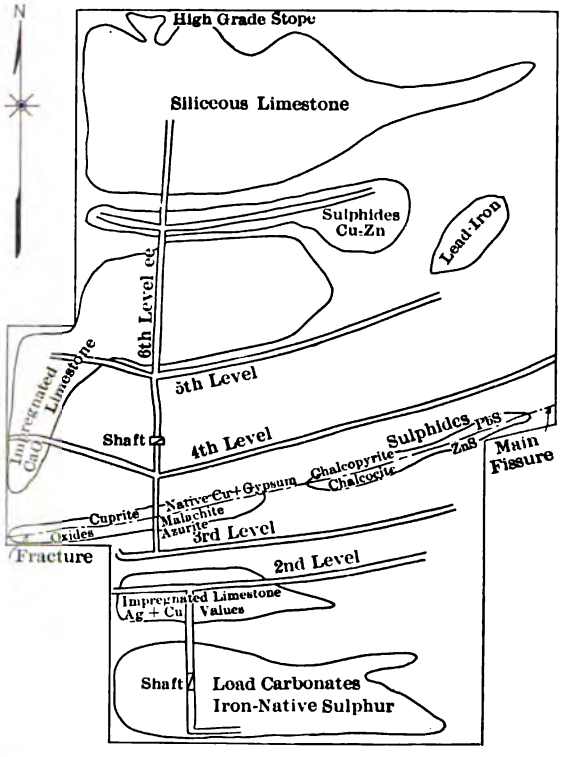
[4] W. Lindgren, Anhydrite as a Gangue Mineral, *Economic Geology*, vol. v., No. 6, p. 522 (Sept., 1910).



(Modification of drawing furnished by R. B. Cochran.)

FIG. 1.—VERTICAL SECTION IN A NORTH-SOUTH DIRECTION THROUGH VETA RICA AND ADJOINING CLAIMS.

rectly precipitated. The sulphur might have been formed by the action of hydrogen sulphide upon the limestone, or on sulphur dioxide, or by the reduction of the gypsum. The most interesting feature about this occurrence is that the gypsum and sulphur, which in other regions have undoubtedly been formed, but either dissolved or carried farther, are here precipitated. This is possibly due to the general aridity of the climate and consequent lack of large amounts of underground water. In



(Sketched by R. B. Cochran.)

FIG. 2.—GROUND-PLAN OF VETA RICA ORE-BODIES.

this ore-body, the limestone foot-wall was not enriched, and the ore was very soft and required square sets in the stopes.

On the 2d level of the south shaft, a drift north through the limestone to the breccia-contact found a body of ore which consisted of a siliceous limestone impregnated with native silver, cerargyrite (var. embolite), and copper-values up to 3 per cent. At the extreme northern part of this deposit, the ore gave

out, and a fracture was discovered, which is locally called the "main fissure" (see Fig. 2). In order to reach this more easily, and also to prospect the northward-dipping contact, the Veta Rica central shaft was sunk. At the 3d level, a drift was run to the south until both contact and fissure were encountered. Ore was discovered at both places, and that found along the contact was of the same general nature as the ore mined on the 2d level of the south shaft, but was of poorer grade. It continued down to the 4th level, slightly below which the shaft cuts through the contact of breccia and limestone.

The ore-body on the "main fissure" was situated about 20 m. from the contact and was found only between the 3d and 4th levels, below which mineralization did not occur. This "fissure" is a break in the limestone nearly at right angles to the direction of dip of the contact-plane, but seems not to have displaced the latter to any appreciable extent. The plane of the fissure also dips northward, but at a much steeper angle than the breccia-limestone contact. The ores on this fissure are richer in copper than any yet mentioned, and are also argentiferous. There are two such silver-copper shoots at this horizon, but one is composed of sulphides, while the other is oxidized (see Fig. 2). These bodies are separated by 120 m. of barren limestone, although both are on the fissure. The sulphide ores are found on the eastern side of the claim and consist of chalcocite and chalcopyrite, with small amounts of covellite, chiefly as tarnish. Some galena and sphalerite are also said to occur here. The chalcocite is all of the gray, massive variety and not the sooty kind which is recognized as secondary in the mines at Butte and elsewhere. On the other hand, the fact that the fissure intersects the breccia-limestone contact and that mineralization extends only to the 4th level seems to indicate that the deposit was made by later, descending solutions, and resulted from secondary precipitation.

The oxidized body is on the west side of the claim and consists of native copper, cuprite, azurite, and malachite, all mixed more or less with gypsum. It is very evident that the solutions which caused the oxidation of this body of ore were either deprived of their oxidizing power before reaching the sulphide shoot on the east or were entirely prevented from contact with it, although both are on the same fracture. The officials of

the mine believe that a water-channel followed down the fissure on the west side, and was prevented from reaching the sulphides by the wall of barren limestone 120 m. thick. As was the case in the lead-deposits of the south shaft, the gypsum is a prominent and interesting feature of the oxide shoot. It was probably formed in the decomposition of copper sulphate according to the following reaction:

$$2CuSO_4 + 2H_2Ca(CO_3)_2 + 3H_2O = \underset{\text{(Malachite)}}{CuCO_3 \cdot Cu(OH)_2} + \underset{\text{(Gypsum)}}{2(CaSO_4 \cdot 2H_2O)} + 3CO_2$$

For the formation of azurite, the following reaction may have occurred:

$$3CuSO_4 + 3H_2Ca(CO_3)_2 + 4H_2O = \underset{\text{(Azurite)}}{2CuCO_3 \cdot Cu(OH)_2} + \underset{\text{(Gypsum)}}{3(CaSO_4 \cdot 2H_2O)} + 4CO_2$$

The cuprite and copper might have been formed by the reducing action of ferrous sulphate, after the manner suggested by Ries,[5] as follows:

$$2CuSO_4 + 2FeSO_4 + H_2O = \underset{\text{(Cuprite)}}{Cu_2O} + Fe_2(SO_4)_3 + H_2SO_4$$

$$Cu_2O + H_2SO_4 = Cu + CuSO_4 + H_2O$$

Since, however, the copper is found with gypsum like malachite, I see no reason why the same reaction which formed malachite and gypsum should not also form copper and gypsum in the presence of some reducing-agent. Any superfluous oxygen might then oxidize some of the copper to cuprite. Some of the specimens bear out this conclusion, as the cuprite is seen in small amounts on the exterior of copper crystals. In still other cases, however, it is mixed with malachite and gypsum. Of course it is also possible that both cuprite and copper might be formed by the reduction of malachite, but the copper, especially, occurs in well-defined cubes imbedded in gypsum, and could not possibly be a pseudomorph formed by the alteration of malachite.

On the north side of the central shaft, between the 5th and 6th levels, two of the largest ore-bodies in the mine were discovered. Here the contact between breccia and limestone becomes perceptibly flatter, and in this respect does not conform

[5] H. Ries, *Economic Geology*, 3d ed., p. 339 (1910).

to usual fault-planes. Of course it is not absolutely certain that a fault does exist between the two rocks, although many facts point to that conclusion. The ore-body nearest the shaft resembles that on the "main fissure" between the 3d and 4th levels, in that it consists of a sulphide and an oxidized portion. As above, the sulphides are on the east and consist of copper-, iron-, and zinc-minerals, while on the west the ores are oxides and carbonates of copper with native silver and cerargyrite (var. embolite). I saw no specimens from this ore-body, but it is to be noted that the two sulphide bodies are both on the east, and that on the west side of the mine the same horizons present oxides. These two bodies, with one other small occurrence, are the only sulphides found in the mine.

Farther northward, but also between the 5th and 6th levels, is the large body of the type forming the largest tonnage of the mine, a "siliceous silver-lime" with small amounts of copper. The composition of this ore ranges around the following values: Ag, 30 oz.; Cu, 0.6 to 3 per cent.; CaO, 20; SiO_2, 22; FeO, 2.5; and barite up to 10 per cent. The ore occurs in the limestone, below the breccia-contact. Although there was ore on the contact, it was not so good as below, and was said to consist of "an argentiferous earthy gangue containing considerable barite." The specimens of the chief ore below the contact which were seen by me consisted of red granular limestone considerably fractured. Many of the pieces showed striated surfaces resembling rough slickensides. Under the microscope, the rock consists of coarser grains of calcite, mixed with finer ones which seem to be the result of grinding, and some quartz. The cracks between the larger particles are filled with embolite and some silver. There is no indication of any replacement, and the ore must be considered as an impregnation of fractured limestone. The solutions evidently contained some iron, which has produced the reddish stain. Although the ore is said to contain workable amounts of copper, no copper-minerals were recognized.

In 1906, during the progress of work in the northern portion of the "silver-lime" very near the north boundary of the claim, a fault was encountered. This is known as the Alvarez fault (see Fig. 1), and has a throw of 40 ft. The fault-plane dips strongly to the south, and must intersect the "main fissure"

at no great depth below the present workings. After working down to the limestone-contact, it was found that the character of the ores had completely changed, and rich silver-copper ores were found in fractured grayish-blue limestone. In places there is a great deal of barite present. This was known as the "high-grade stope;" the ores often ran up to 150 oz. of silver per ton, with some copper. The minerals found here were native silver, argentite, proustite, pearceite, and erythrite. The pearceite, $(Ag, Cu)_{16}As_2S_{11}$, which is arsenical polybasite, has been found previously at but four other localities in the world: Schemnitz, Hungary; Arqueros, Chile; Aspen, Colo.; and Marysville, Mont. The Veta Rica pearceite contained 15.65 per cent. of copper, but no antimony. Tests on the argentite and proustite show that both contain copper, and indicate that the solutions which deposited this ore contained chiefly silver, copper, arsenic, and sulphur. Thin sections of one of the specimens show a mass of lamellar, plumose barite impregnated with argentite, and silver. In most places the silver surrounds the argentite, and was probably formed by the reduction of the latter. In a few instances, small grains of silver were coated with a black film of argentite. If the native silver is excepted, the only common oxidation-product found here is the erythrite, which often occurs in well-defined crystals in cavities and cracks in barite. This is the first occurrence of cobalt-minerals which has been observed in Sierra Mojada. It seems probable that cobalt arsenides are present in some of the ores, but they have not been detected. Since this particular association of erythrite and barite was said to contain more than 100 oz. of silver per ton, the erythrite was carefully examined and was found to be coated in places with exceedingly fine hairs of native silver, which sometimes accumulated into globular masses. In all cases, the silver was deposited on the erythrite and was therefore formed later. I have no means of ascertaining what relation this association had to the occurrence of silver with argentite, proustite, and pearceite. However, it would seem that in both associations native silver was one of the most recent minerals formed.

VII. Paragenesis and Origin of the Ores.

In the description of the various ore-bodies, the association of the different minerals has been given, and where possible,

attention has been called to the order of their deposition. Since, however, most of the specimens at my disposal are in separate pieces, and nothing is known of their relations to each other, no definite conclusion can be reached on the order of succession of the various minerals. This can only be determined in the mine itself. There a distinct association seems to be shown between silver and copper on one hand, and between silver and lead on the other. This would conform to the intermediate solubility of silver sulphate when compared with lead and copper sulphates. The association of lead and silver was seen on the 1st level of the south shaft, and is apparently the most common one in the district. It was likewise the only one known from 1878 until 1893, when the first copper-ores were discovered. Since most of the ores are oxidized, very little zinc is found with them—in accordance with the greater solubility of zinc sulphate as compared with the sulphates of copper, silver, and lead. In one or two mines a little earthy zinc carbonate has been found below lead carbonate ores in the limestone below the contact. In several mines, lead carbonate ore containing silver was found below the copper-silver ores. This was especially true in the San José mine, and seems to indicate that a migration of solutions had occurred in which the lead-silver ores had wandered farther than those containing copper-silver—a phenomenon which does not agree with the relative solubility of their sulphates. This might be explained by the presence of certain compounds which would render the lead more soluble, or by the assumption that two different solutions deposited the ores at two different periods.

In regard to the origin of the ores, the data at my disposal are not sufficient to warrant any conclusion. It is, however, very clear that there is a definite relation between the deposition of ore, and the plane of contact between breccia and limestone, whether this be a fault-plane, or an ore-channel between two dissimilar rocks. All ores are found below the breccia, which is considered quite impervious, and therefore would not be much affected by either ascending or descending solutions. Since most of the ores are oxidation-products, and therefore secondary, the question of origin evidently depends upon the interpretation of the three sulphide bodies found in the mine. These are either parts of an original deposition,

along the contact of limestone and breccia, and in the "main fissure," which has resisted oxidation, or they have been precipitated after secondary concentration. The various minerals found in the sulphide ores, such as chalcocite, chalcopyrite, argentite, proustite, pearceite (polybasite), and silver, according to the writings of Lindgren[6] and W. H. Emmons,[7] are possible in primary deposits as well as secondary enrichments. Neither are there in the Veta Rica deposits any convincing criteria of downward enrichment as suggested by Ransome.[8] According to Sales,[9] gray massive chalcocite, such as occurs here, is strongly indicative of primary origin. The presence of sulphide ores in the "main fissure," about 20 m. from the contact of breccia and limestone, and the fact that they are found only between the 3d and 4th levels, rather indicate descending solutions. In regard to the "silver-lime" ores which are found below the contact, as impregnations of cracks in the limestone, the indications are that the deposits were probably formed by descending waters which had leached the material from the contact-plane above. The presence of considerable cerargyrite (var. embolite) seems to be rather characteristic of arid regions,[10] and is believed by Keyes[11] to have been largely due to wind-blown saline dust, taken up by meteoric water, which causes the alteration of ore-bodies, with the consequent formation of silver chloride. If, however, the observation is correct that the Sierra Mojada ores were formed before the deposition of the conglomerate and the younger Cretaceous limestone, it is evident that the action of percolating sea-water would have tended to cause the formation of silver chloride long before the present arid conditions prevailed. The presence of small amounts of

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[6] W. Lindgren, The Relation of Ore-Deposition to Physical Conditions, *Economic Geology*, vol. ii., No. 2, p. 105 (Mar.–Apr., 1907).

[7] W. H. Emmons, A Genetic Classification of Minerals, *Economic Geology*, vol. iii., No. 7, p. 611 (Oct.–Nov., 1908).

[8] F. L. Ransome, Criteria of Downward Sulphide Enrichment, *Economic Geology*, vol. v., No. 3, p. 205 (Apr., 1910).

[9] R. H. Sales, Discussion of Criteria of Downward Sulphide Enrichment, *Economic Geology*, vol. v., No. 7, p. 681 (Oct.–Nov., 1910).

[10] R. A. F. Penrose, Jr., Superficial Alteration of Ore-Deposits, *Journal of Geology*, vol. ii., No. 3, p. 314 (Apr.–May, 1894).

[11] C. R. Keyes, Cerargyritic Ores: Their Genesis and Geology, *Economic Geology*, vol. ii., No. 8, p. 774 (Dec., 1907).

bromide in the silver chloride might be accounted for in this manner.

VIII. Conclusion.

I frankly admit that I have been unable to arrive at any convincing conclusion regarding certain geological features presented in this district. Most evidence points to the fact that the valley was made by faulting and not by the erosion of an anticline, and that the ores were deposited in a fault-plane or ore-channel between breccia and limestone. All this occurred before the deposition of the younger Cretaceous limestone. Since most of the ores are now oxidized, it seems most probable that the few sulphide bodies are portions of the original deposition, and have not been formed by secondary concentration.

I trust that I have been able to state a few facts which were not previously known about a region which is of import economically, and interesting geologically. It is to be hoped that any one who may visit the district will pay especial attention to the various features which are still open to discussion, such as the formation of the valley and the origin of the breccia.

Geology of Harrison Gulch, in Shasta County, California.

BY H. E. KRAMM, ITHACA, N. Y.

(New York Meeting, February, 1912.)

During the summer of 1910, I had the opportunity to study in detail the geological conditions of what is known in northern California as "Harrison gulch," in Shasta county. Though the district, as a whole, that is, the western part of Shasta and the eastern part of Trinity county, has contributed during the last 15 years a considerable share of the gold-production credited to Shasta county, very little is known of its geology, perhaps because the country is covered with a dense growth of brush which makes exploration and prospecting extremely difficult.

Topographically the district presents a succession of ridges trending approximately N–S., and alternating with deep, narrow canyons. The drainage is partly into the Sacramento, partly into the Trinity river.

Climatic conditions in summer time are almost ideal. It is true, the thermometer may climb to 115°, but the air is dry

and the heat is bearable, especially as the nights are always cool. It seldom rains during the summer months, and creeks and springs dry out towards the end of the summer. The winter is severe, lasting from November to April, and usually a thick layer of snow covers the mountains.

By reason of the character of its gold-deposits, the region is known to miners as "the pocket country." It contains a series of eruptive rocks intruded into slates, cherts, limestones, and conglomerates. Dikes, laccoliths, and batholiths are frequent. In a day's ride it is possible to find all the common igneous rocks from the very acid to the very basic. Serpentines are especially abundant. The sedimentary rocks are, as a rule, broken and shattered as well as folded, and show the effects of intense metamorphism. "Pocket-hunters" speak of the contact of diabase and slate as the "pocket-contact." Some rich gold-pockets, however, have also been found in the diabase.

Besides the gold, there are indications of quicksilver and copper. Cinnabar is often present in the concentrates of the placer-gravels, and can invariably be traced to a contact of serpentine and shale. It seems, therefore, that the conditions of its occurrence are similar to those of the central coast ranges of California, where, with a few exceptions, it is found in decomposed serpentine near the contact of that rock with shale or sandstone.

Up to the present time quicksilver has not been found in paying quantities in Shasta county, but such a discovery would not be surprising.

Indications of copper are stains of malachite and chalcopyrite. They are, in all probability, due to the leaching of copper-minerals which most basic eruptive rocks carry, and which are redeposited. They are especially abundant in quartz veins found in slates and basic eruptive rocks; and since the quartz may also be a carrier of gold, the presence of the stains may make the cyanidation of such ore a difficult problem.

The area described in detail here comprises about 6 sq. miles, including that part of Harrison gulch in which the property of the Goldhill Consolidated Mines is located, and also the town of Knob.

Knob is reached from Redding, which lies to the northeast,

by daily stage, which leaves at 6 a. m., passes through Ono and Igo, and in summer time manages to reach its destination in the evening. A road also makes connection with Red Bluff. Trails lead to Weaverville in the north and to other points. Redding and, to some extent, Red Bluff are bases of supplies.

Figs. 1 and 2 give an idea of the topography and geology. Harrison Gulch creek, which empties into Cottonwood creek, a tributary of the Sacramento river, has its headwaters within

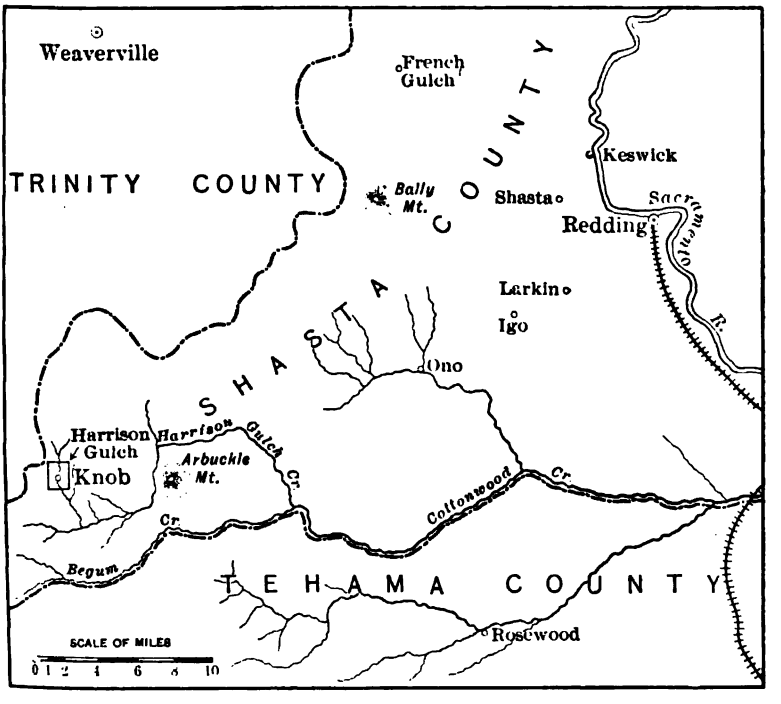

FIG. 1.—SKETCH-MAP OF SHASTA COUNTY, CAL.

the area. Its course is approximately N–S., and it has cut a steep canyon, the walls of which rise in places more than 1,000 ft. above the creek-bed. Numerous smaller streams have left their imprint upon the topography, the rugged appearance of which is heightened by the dense growth of brush and an abundance of pine and fir timber.

The following rocks appear in this district:

1. A shale of siliceous character, reddish yellow in color, fairly hard, and showing a pronounced cleavage. Near its con-

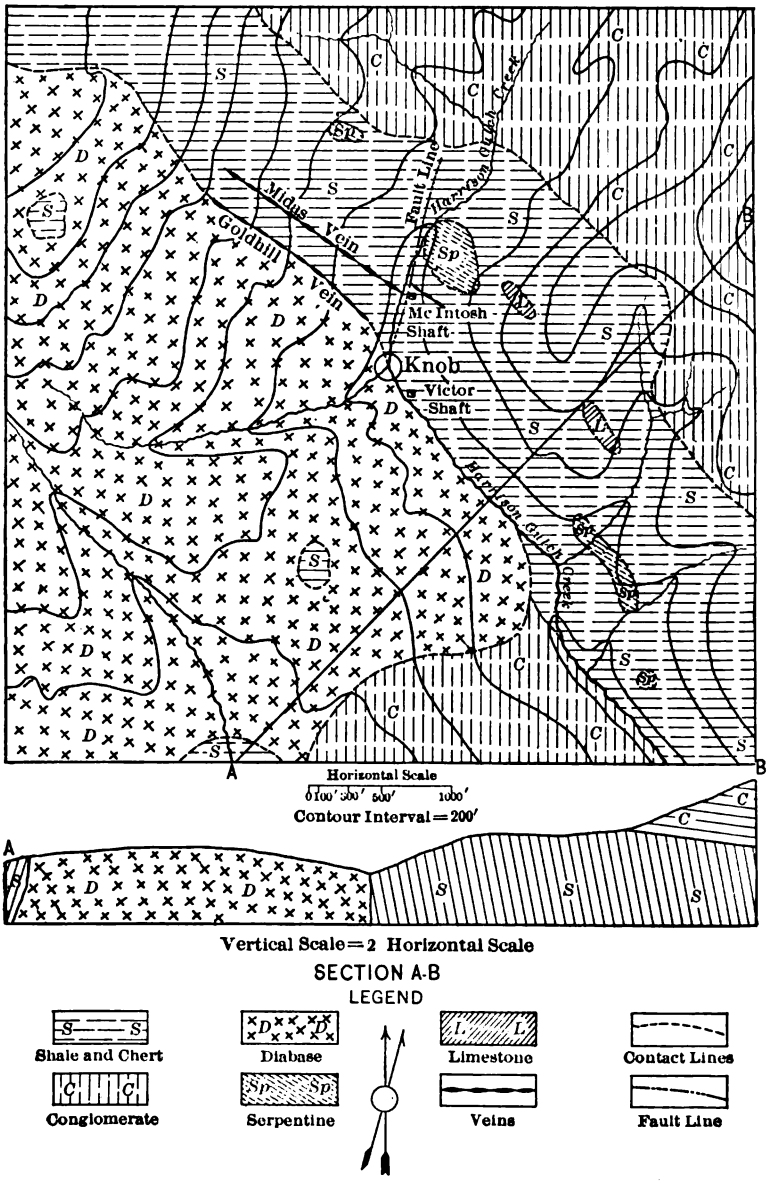

FIG. 2.—GEOLOGY AND TOPOGRAPHY OF HARRISON GULCH, SHASTA COUNTY, CAL.

tact with a diabase, which is intrusive, it becomes slate, its color changes to a greenish brown, and it carries considerable amounts of free carbon. Interbedded with it are a more or less iron-stained chert and lenticular bodies of crystalline limestone of great purity. The former appears to represent an intense degree of metamorphism of the shale.

2. A conglomerate, made up of pebbles of chert which vary in size from a diameter of several inches to that of a grain of sand. It is well cemented by a siliceous material. A friable shale, interbedded with the conglomerate, contains, near Knob Hill, north of Knob, a fossil flora.

3. Serpentine derived from a peridotite which is intrusive into the shale first named above. It ranges in color from dark green to brown, according to its state of decomposition. As a general rule, it is fairly well preserved, and shows several of the original minerals of the peridotite still intact.

4. A diabase which seems to grade on one hand into a gabbro, and on the other into an andesite. It is somewhat decomposed and intrusive into the shale and conglomerate.

The shale, limestone, and chert were deposited first. Their angle of dip is high—approximately 70°. An unconformity exists with the overlying conglomerate.

The area investigated gives no clue as to the geological age of the rocks. They seem, however, to correspond in every respect with the Clear Creek volcanic series and the Bragdon formation, consisting of slates, quartzites, and conglomerates.

Geology of the Deposits.

The intrusion of the eruptives has given rise to conditions favorable to the formation of quartz fissure-veins. Stringers can be found traversing the slate and the diabase at many places. The two important veins mined for gold are known as the Midas and the Gold Hill.

For sake of description we may divide all the quartz veins of the area into three types.

The first type is represented by the Midas vein, a fissure in the slate, which has been worked to a depth of about 1,500 ft. It strikes approximately NW–SE., and dips about 80° S. Its width is variable, with a maximum of 3 ft. The walls are composed of broken and shattered minute angular fragments

of slate, recemented by chalcedony. This condition is not confined to the wall-surface alone, but extends a considerable distance into the rock. It is scarcely visible to the naked eye, but can plainly be seen under the microscope.

The ore-minerals consist of native gold (alloyed with some silver), iron pyrites, and tetrahedrite. A thin gouge is sometimes present; more often the quartz rests directly on the slate wall.

The ore-body has undergone extensive faulting. Besides numerous faults of minor importance, among which there seems to be no system, there are two of considerable magnitude.

The first is known as the "header and gutter"—a horizontal displacement, averaging about 5 ft., which can be traced from level to level without serious difficulty, and which is important because it has been instrumental in the concentration of the gold. Below it, the values decrease rapidly; above it, they are high.

The second fault is more extensive. It is a normal fault striking approximately N–S.—that is, nearly at right angles to the strike of the vein. Its course is about that of Harrison Gulch creek near the McIntosh shaft. It dips 80° E. This fault caused a downward movement of 200 ft. of the country east of the creek, with a horizontal displacement to the north of about 200 ft. It is easily recognized and traced. Its walls at places stand 6 ft. apart, the space being filled with fault-breccia, gouge, and broken fragments of the ore-body. At other places the walls touch each other.

The "header and gutter" has been found to continue in that portion of the vein which was displaced by the second fault.

The second type of veins is represented by the Gold Hill, which differs from the Midas in that it is located along the shear-zone caused by the intrusion of diabase into the slate. It is irregular in dip and strike and eccentric in metal-content. A gouge of talc is often, but not always, present. Besides the ore-minerals found in the Midas vein, chalcopyrite in small quantities and native copper are present, the latter, however, being exceedingly rare.

Where no vein exists between the two rocks, the contact is sometimes sharply defined. On one hand is slate, on the

other diabase, with a definite boundary-line. Sometimes, however, it is not well defined, but one rock so gradually passes into the other that even the microscope does not give satisfactory information as to the mineral content, the rock being much decomposed and showing thermal effects.

The third type embraces veins entirely in diabase. Their origin is probably due to the cooling of the intrusive rock and consequent shrinkage, which caused fissuring. They are very irregular in occurrence and shape. It is of interest that up to the present these veins have proven barren with the exception of those which lead into the shear-zone and contain values near this zone.

The distribution of ore-minerals in the three types of veins makes it certain that the slate was an influential factor in their deposition. Mineral-bearing solutions, meeting the carbonaceous waters issuing from the slate, were forced to deposit their mineral content. This explains why the third type of vein is mineralized near the shear-zone, but barren some distance away from it. It may also explain why the ore-minerals are most evenly distributed in veins of the first type, where both walls are slate and could act as precipitating-agents.

That the carbon acts as a powerful reducing-agent is demonstrated by the occurrence of native copper along the shear-zone. This carbon has proved a hindrance in the cyanidation of the ore.

In mining the ore, it is not possible to avoid including a certain percentage of slate. The ore is crushed and amalgamated, and the tailings are cyanided. While the percentage of copper present is not enough to interfere with the cyanidation, the free carbon entering with the slate has caused serious difficulty. It was at first believed that the sulphides contained in the ore were the objectionable elements, and roasting was tried with success. As a matter of fact, however, the roasting transformed most of the carbon into carbon dioxide and thus freed the ore of the objectionable reducing-agent.

In closing, I wish to acknowledge my obligation to H. L. Waste, Manager of the Goldhill Consolidated Mines Co., with whose permission this paper has been prepared.

The Geology and Ore-Deposits of the Silverbell Mining-District, Arizona.

BY C. A. STEWART, MOSCOW, IDAHO.

(New York Meeting, February, 1912.*)

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* Presented in oral abstract at a meeting of the Spokane Local Section of the Institute, Nov. 24, 1911.

I. Introduction and General Statement of Results.

The field-work upon which this paper is based was done in the summer of 1910, and was made possible by the courtesy of the Imperial Copper Co., which gave me access to its property. For their kindness in this respect I wish to express my thanks to W. F. Staunton and Meade Goodloe. Moreover, B. F. Smith and G. B. Gentry, mining engineers in the employ of the company, gave invaluable assistance by placing maps, etc., at my disposal, and by discussing with me the problems of the district; and it is a great pleasure to acknowledge this indebtedness to them. In order that the significance of some of the details that follow may be better appreciated at first reading, a general outline of the conclusions reached is given here.

In the Silverbell mining-district a series of Palæozoic limestone blocks is completely surrounded by post-Palæozoic igneous rocks, intruded in the following order: alaskite, alaskite-porphyry, biotite-granite, andesite, and quartz-porphyry. The biotite-granite is believed to represent a differentiation of the magma from which the alaskite-porphyry came; and this conclusion has an important bearing on the origin of the ores. The intrusion of both alaskite-porphyry and biotite-granite was followed by the emission of magmatic waters, which sericitized and silicified the alaskite-porphyry and granite, and produced in the limestone, by the addition to it of silica, iron, and alumina, great masses of garnet, quartz, and wollastonite. Following close upon these solutions came metal-bearing magmatic waters, which impregnated porphyry, granite, and alaskite with cupriferous pyrite, and deposited in the garnet zones chalcopyrite and copper-bearing pyrite that make important bodies of contact-metamorphic ores. Secondary enrichment has taken place in the disseminated ores in the igneous rocks, raising their copper-content to 2 per cent. in some cases, and extensive drilling has been undertaken to block out these deposits. East of the contact-metamorphic deposits there is a fissure-vein of lead-silver ore in the quartz-porphyry. This rock is distinctly younger than the alaskite-porphyry and the granite; and the lead-silver ores belong, therefore, to a period of mineralization distinctly later than the copper-deposition.

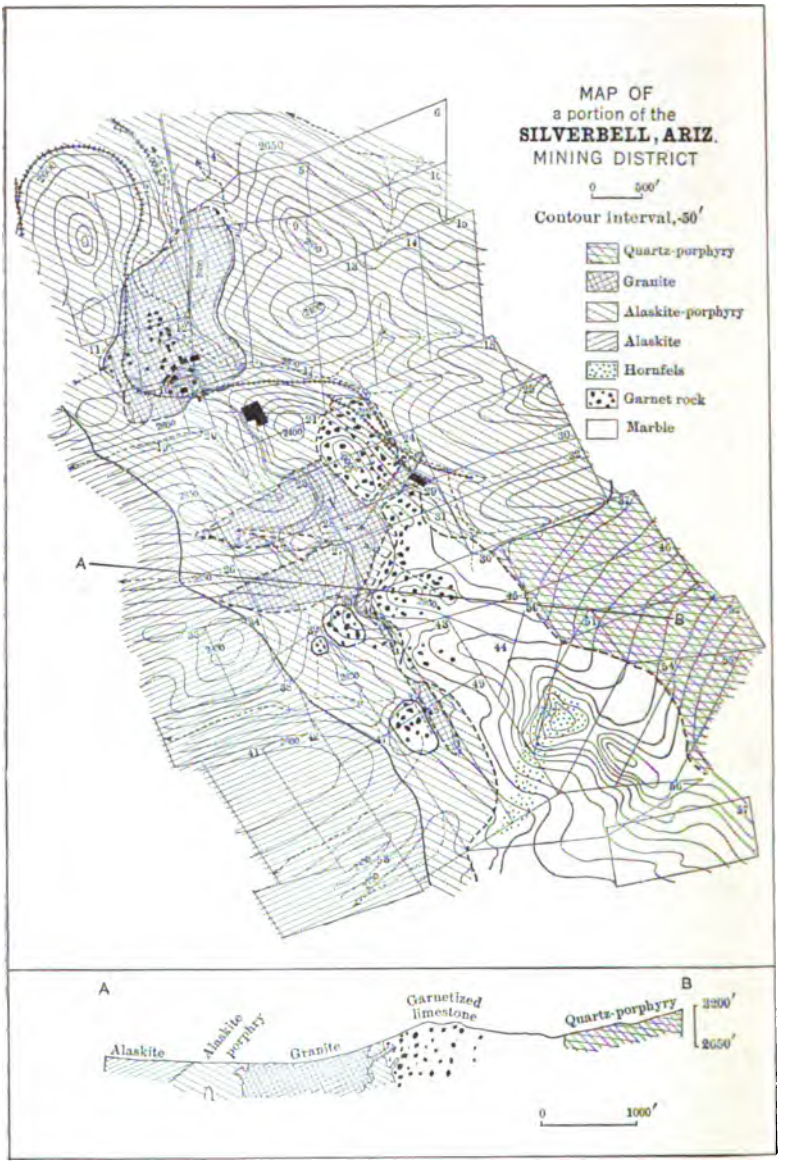

FIG. 1.—MAP OF A PORTION OF THE SILVERBELL MINING-DISTRICT, WITH GENERALIZED SECTION ON LINE A–B.

KEY TO CLAIMS AS NUMBERED IN FIG. 1.

1 Jesuit.
2 Pope.
3 Prince of Wales.
4 Columbia.
5 Ruby.
6 Bobby.
7 Peerless.
8 Belle.
9 John F.
10 Eugene.
11 Daisy.
12 Leslie.
13 Billy.
14 Klondike.
15 Northern Star.
16 Wilson.
17 Black Eagle.
18 Red Ridge.
19 Bay State.
20 Maggie.
21 Wedge.
22 Imperial.
23 Florence.
24 Southern Beauty.
25 Enterprise.
26 Hilda.
27 Yankee.
28 Spike.
29 Union.
30 Northern.
31 Comet.
32 Alliance.
33 Oversight.
34 Frank B.
35 Hamilton.
36 Page.
37 Protection.
38 Detroit.
39 Sampson.
40 Mammoth.
41 Taft.
42 Olympia.
43 Mountain View.
44 Wedge.
45 Fraction.
46 Chance.
47 Queen.
48 Prospector.
49 Emerald.
50 Confidence.
51 Silverbell.
52 Long Shot.
53 Pima.
54 Black Daisy.
55 Herbert.
56 Mollie.
57 Murray.

The chief points of general interest in the district are the peculiar relationships existing between the alaskite-porphyry and the granite; the occurrence of quartz dikes; the shape of the garnet masses, which suggests formation by some process other than replacement; chemical evidence, confirming the view that contact-metamorphic garnets are not usually the result of recrystallization; and the establishment of a rather definite time for the emission of magmatic ore-bearing solutions, which have continued after the intrusion. Of economic importance are the establishment of the distinctly later age of the lead-silver deposits; the recognition of the importance of both granite and alaskite in the formation of the contact-ores; and the interpretation of some of the facts found in prospecting the disseminated deposits.

II. Location, Topography, and History.

The Silverbell mining-district is situated in the western part of the Silverbell mountains, in Pima county, Arizona, about 40 miles west of Tucson, and 20 miles south of Red Rock, on the main line of the Southern Pacific railroad, with which station it is connected by the Arizona Southern railroad, a standard-gauge road built and controlled by the Imperial Copper Co.

The Silverbell mountains form the northern spur of the Roskruge range, one of that series of low but rugged mountain-chains with NE–SW. trend which rise sharply out of the flat, barren wastes of the Arizona desert region. These mountains are the stumps of larger hills which have been cut down by erosion, the detritus of this erosion having in part filled the intervening valleys and produced the level plains of the desert. Elevations in the Silverbell district vary from 2,500 to 4,000 ft.; and the climate, although somewhat cooler than that of the sandy stretches around Red Rock, is dry and hot. The vegetation is typical of the desert—mesquite and various forms of cacti predominating.

Within the Silverbell district itself the topography is very rugged, as indicated on the map, Fig. 1. The elevations on this map were determined with an aneroid and are therefore approximate. The topography is also shown in Figs. 2, 3, 4, and 5.

In general, the mountains of the Arizona desert region are similar to the Basin ranges of Gilbert,[1] which represent lines of monoclinal or, more rarely, anticlinal folding.[2] In the limited area covered by me, however, only late intrusive rocks with a few detached blocks of sediments were seen; and it was impossible to gather any data as to the mountain structure. It is therefore uncertain whether the Silverbell mountains represent an intrusive area in a typical Basin range, or whether the whole Roskruge is the remnant of a series of resistant igneous rocks.

For nearly 50 years, prospectors have been attracted to this region by the heavy, black, copper-stained garnet outcrops. In the early days several small but rich pockets of lead-silver ores were opened up. These were, however, soon exhausted, and intermittent attempts were made to work the copper-ores associated with them, but with no success until 1903.

At this time the Imperial Copper Co. was organized as successor to the Silverbell Mining Co., and began systematic mining and development on its property of some 60 claims, which included the Old Boot mine, first opened in 1865. A railroad was built to Red Rock; in 1904, more than 3,000,000 lb. of copper was produced; and this amount increased steadily to 1909, when the total production was 11,000,000 lb. The Old Boot mine has been renamed the Mammoth, and now has a 700-ft. vertical and a 900-ft. inclined shaft, Fig. 6. Two other mines, the Union, with a 450-ft. vertical shaft, Fig. 7, and the Billy, 350 ft. deep, have also been opened.

At first the ore was shipped to the Copper Queen works at Douglas for reduction; but in 1908, blast-furnaces and converters capable of handling 800 tons of ore a day were completed at Sasco, on the Arizona Southern railroad, 5 miles north of Silverbell.

In August, 1910, the company stopped mining and shut down the smelter, because of the depression and uncertainty in the copper industry. It is, however, continuing development-work in the contact-ores, and churn-drill exploration of the disseminated deposits.

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[1] Ransome, F. L., *Professional Paper No.* 12, *U. S. Geological Survey* (1903).

[2] Wheeler's *Report on Geographical and Geological Surveys West of the One Hundredth Meridian*, vol. iii. (1875).

III. Description of the Rocks.

1. *Altered Sediments.*

Distribution.—Limestone, now changed to marble, and some peculiar siliceous rocks, considered as altered impure phases of the same formation, are found along a NW–SE. line, in irregular blocks completely surrounded by igneous rocks. The shape and distribution are shown in Fig. 8. The largest block is nearly 2 miles long, and has a maximum width of more than 2,000 ft. This "lime belt," as it is locally called, forms a steep ridge, because of the resistance offered to erosion by the peculiar siliceous rock (described below), or by the garnetized portions. No other sediments were found in the region visited; and these isolated blocks of marble appear more like fragments swamped in an intrusive than like members of a sedimentary series in place.

Description.—The typical marble is white or white with gray bands and coarsely crystalline, the grains averaging 0.25 in. in diameter. Except in the garnet zones it is fairly pure, and in it are found tabular masses of black marble often several hundred feet long, which represent beds of more carbonaceous material. The complete destruction of all continuous bedding, and the general "stewing up" that the whole mass has undergone, are shown by the irregular way in which these black patches are scattered through the white marble. Knots of quartz, probably formerly flint nodules, and stringers and lenses of finer-grained, more siliceous limestone are sometimes found.

Capping the ridge in the Silverbell and Black Daisy claims, and forming a buttress on the SW. side of the hill, is a puzzling rock, thought in the field to be a rhyolite, but appearing upon further study to be the result of recrystallization and silicification of a clay-quartz bed in the original sediments. In the hand-specimen this rock is gray, blue-gray, or greenish, exceedingly tough and hard, breaks with a conchoidal fracture, and is as fine-grained as a novaculite. On the weathered surface, it sometimes shows a parallel streaking that might be flow-structure. Microscopically, the rock is a very fine mosaic of quartz and possibly feldspar, with varying amounts of chlorite, epidote, and diopside, and occasional garnet, Fig. 9. The

chlorite, diopside, and epidote are always microscopic in size, often too small for positive identification. They may be only accessories, or may make up the greater part of the rock, and, when arranged in bands, account for the "flow-line" appearance of the weathered material. This rock, found in irregular masses in the underground workings, is known to the miners as "altered lime." At the surface it often shows a dark brown or black "varnish," and, because of its resistance, stands out like the garnet rock, for which it is mistaken from a distance. Although lacking the knotted, spotted appearance of many rocks formed by the metamorphism of shales, this rock is best classed as a hornfels, and will be spoken of as such hereafter.

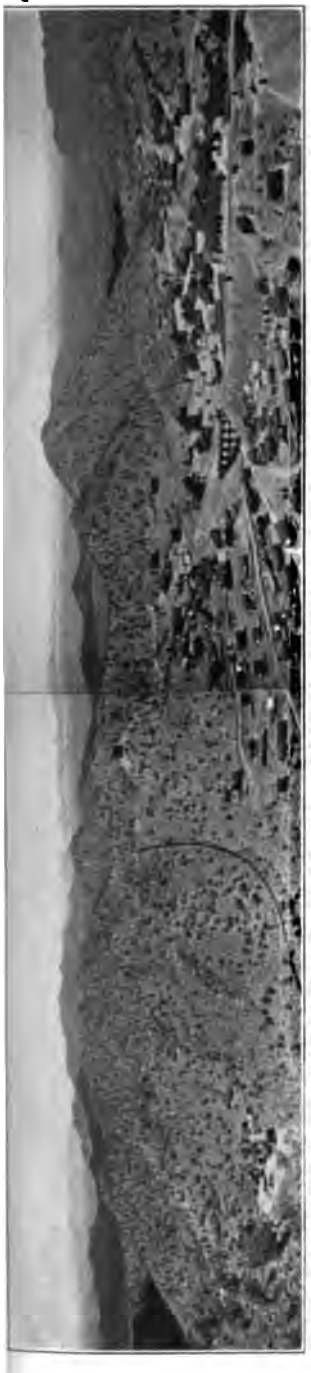
Age.—No determinable fossils were found in these rocks, but there were seen in the marble some bits of clearer calcite which, from their shape, were undoubtedly remains of organisms. To a vivid imagination, the general outline, in some instances, suggested brachiopods. Evidently basing his statement upon knowledge of the surrounding country, W. P. Blake has called this marble Carboniferous,[3] and the same opinion is held by Prof. C. F. Tolman, Jr.,[4] the present Territorial Geologist, who notes Carboniferous strata in the Silver hills to the north.[5]

2. *Alaskite.*

Distribution.—This rock forms all of the Silverbell mountains southwest of the lime belt, being bounded on the northeast as shown in Fig. 8, and disappearing to the south, west, and north under the detrital material of the desert out of which these mountains rise abruptly. The area thus exposed is roughly 10 square miles.

Description.—The alaskite is a coarse, even-grained rock consisting of quartz and feldspar, with practically no dark silicates. The weathered material seen on the hills is usually dark red, and because of the projection of the knobs of quartz has a rough and jagged surface, with an appearance of excessively coarse grain recognizable from a considerable distance. The

[3] Report of Governor of Arizona for year ended June 30, 1904, pp. 66 to 72.
[4] Private communication.
[5] *Mining and Scientific Press*, vol. xcix., No. 22, pp. 710 to 712 (Nov. 27, 1909).



Alaskite-porphyry.

FIG. 2.—LOOKING NORTH FROM TOP OF HILL IN BAY STATE CLAIM (19).

Alaskite on the west; alaskite-porphyry and granite on the east. This view shows the northern part of the Silverbell mountains rising out of the desert.

Marble and garnet rock.

FIG. 3.—LOOKING NORTHWEST FROM TOP OF RIDGE IN SILVERBELL CLAIM (51).

Note rough outcrops of coarse alaskite on hills to the west; inclusions of garnet rock in alaskite-porphyry; black garnet cappings show in middle distance and at northern end of marble block; jagged rock in center foreground is hornfels.

FIG. 4.—LOOKING EAST IN THE BLACK EAGLE-IMPERIAL CLAIMS (17, 22).

At the right is black garnet rock; lower right is granite-porphyry, and left is alaskite porphyry.

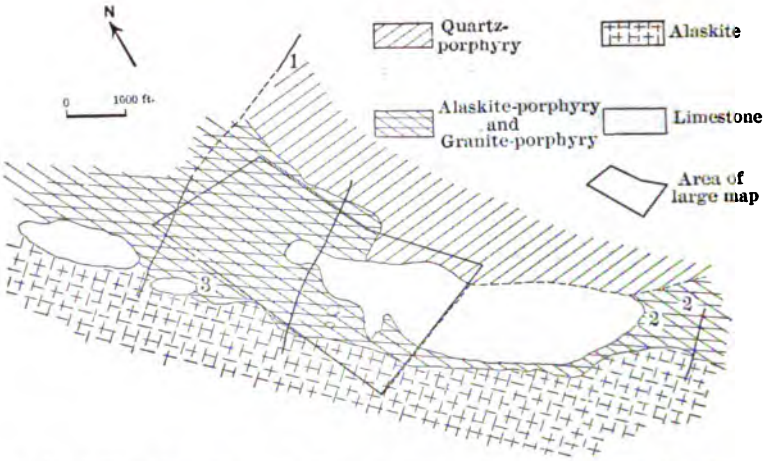
FIG. 5.—PROSPECTING FOR DISSEMINATED ORE IN THE ALASKITE.

The picture shows also the rough character of the hills of this rock.

FIG. 6.—MAMMOTH VERTICAL AND INCLINE SHAFTS.

FIG. 7.—UNION SHAFT AND POWER-HOUSES.

Dark rock in upper left-hand corner is garnetized limestone; depression at right of shaft is dike of granite-porphyry; alaskite-porphyry in foreground, contact with lime covered by dump.



1. Lead-Silver Prospect. 2. Young America Property. 3. El Tiro Shaft.

FIG. 8.—SKETCH-MAP SHOWING GENERAL RELATIONSHIPS IN THE SILVERBELL MINING-DISTRICT.

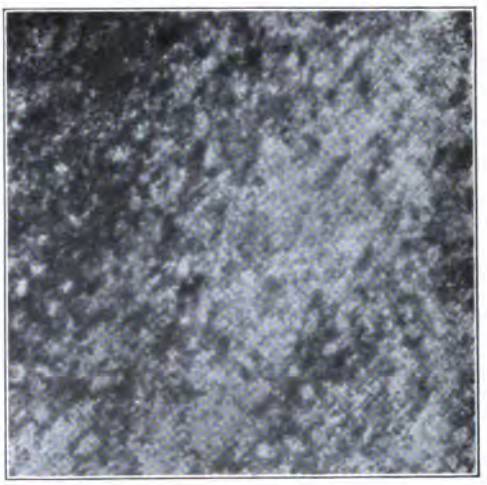


FIG. 9.—HORNFELS. ORDINARY LIGHT. × 100 DIAMETERS.

The white is quartz and probably some feldspar; the gray is chlorite, epidote, and diopside.

FIG. 10.—SILICIFIED ALASKITE-PORPHYRY. CROSSED NICOLS. × 50 DIAMETERS.

The felty mass in the upper left-hand corner is unaltered ground-mass; all of the rest of the slide is secondary quartz.

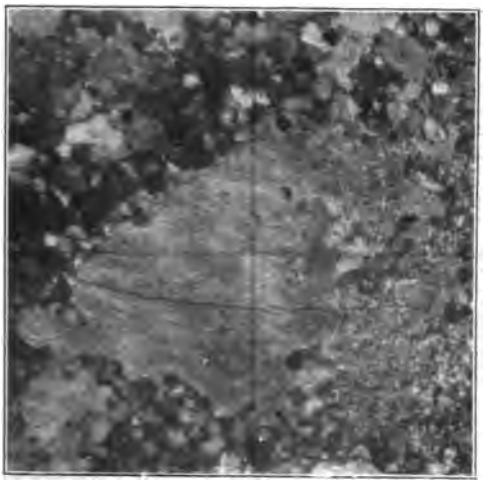


FIG. 11.—SILICIFIED ALASKITE-PORPHYRY. CROSSED NICOLS. × 100 DIAMETERS.

In hand-specimen this rock was a "quartzite," not distinguishable from a specimen of a quartz dike. All of the above picture is secondary quartz except the dusty fragment in the center, which is a feldspar phenocryst in the process of silicification.

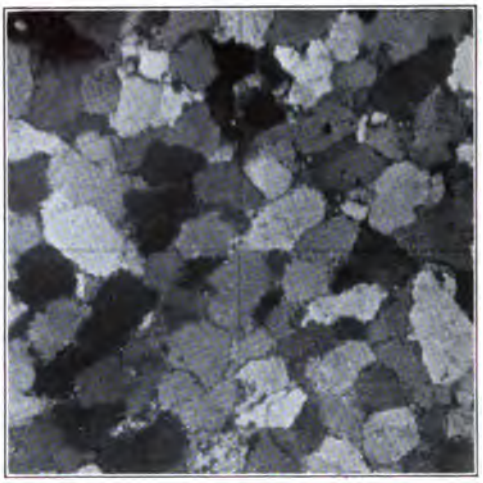


FIG. 12.—QUARTZ DIKE. CROSSED NICOLS. × 50 DIAMETERS.

All of the grains are quartz.

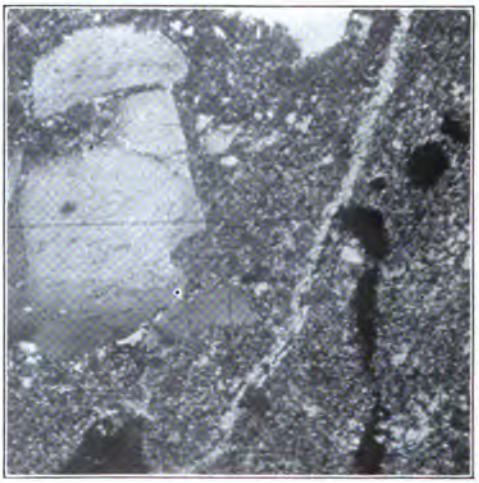


FIG. 13.—ALASKITE-PORPHYRY. CROSSED NICOLS. × 50 DIAMETERS.

This is as near unaltered alaskite-porphyry as was found. The phenocrysts are primary quartz; ground-mass shows some sericite, and a vein of sericite is seen.

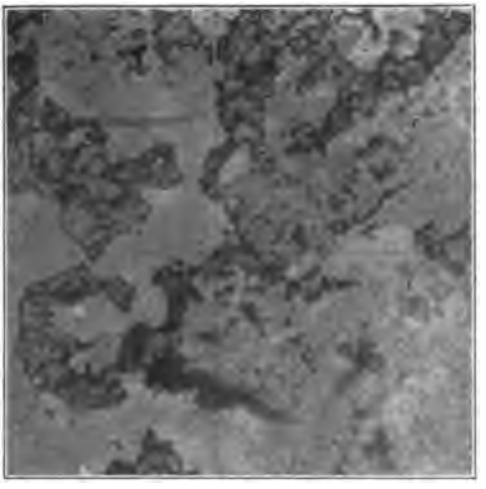


FIG. 14.—GARNET REPLACING ALASKITE-PORPHYRY. ORDINARY LIGHT. × 50 DIAMETERS.

Mineral in high relief is garnet.

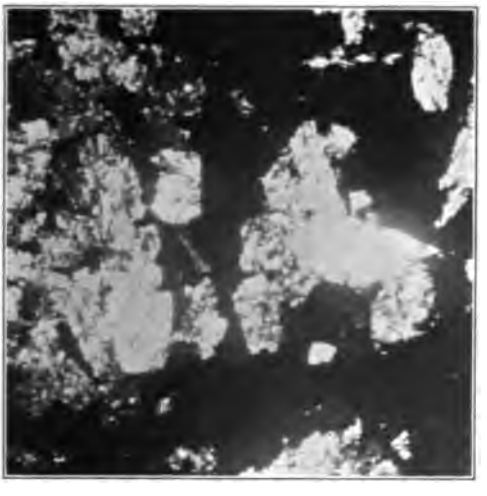


FIG. 15.—GARNET ROCK. CROSSED NICOLS. × 50 DIAMETERS.

Black is garnet; white and gray, wollastonite.

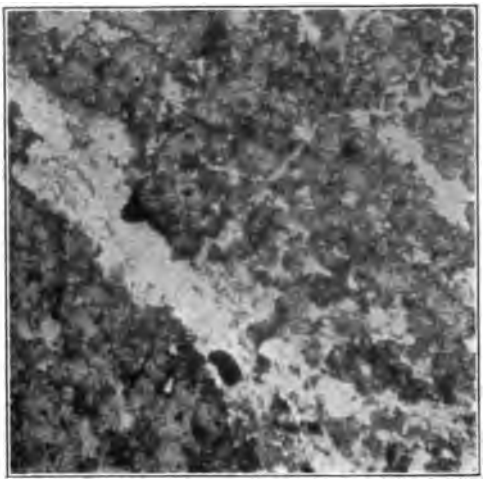


FIG. 16.—LEAN ORE. ORDINARY LIGHT. × 50 DIAMETERS.

Rough material is garnet; white is quartz with a little calcite; the black spots in the veinlet are pyrite. This is the commonest relation of these minerals and shows the later genesis of quartz and ore.

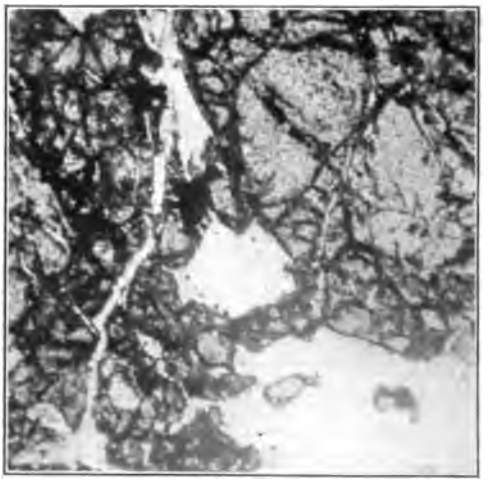


FIG. 17.—GARNET AND LATER QUARTZ. ORDINARY LIGHT. × 50 DIAMETERS.

Note suggestion of porous garnet mass filled with quartz.

fresh rock is light gray, and the actual diameter of the grains is seldom more than 0.5 cm. Microscopic examination shows that the feldspar forms approximately two-thirds of the rock and is largely orthoclase, the subordinate plagioclase being nearly pure albite. Zircon and magnetite (probably titaniferous) are unimportant accessories, and an occasional flake of biotite is seen. The dusty appearance of the quartz is characteristic, and much of the rock shows cupriferous pyrite or its decomposition-products, which will be further discussed under "Disseminated Ores." Some of the gulches showed patches of rock rich in hornblende or biotite; but the general lack of minerals other than quartz and feldspar justifies the name of alaskite.

Of important economic significance are the parallel, nearly vertical joints of a NE–SW. strike that occur rather generally, but still somewhat irregularly, throughout this alaskite, especially along the contact with the alaskite-porphyry. They are not typical prismatic joints of an igneous rock, nor do they seem connected with displacement; they may be due to stresses produced by later intrusives.

Aplite dikes from 1 to 12 in. in thickness, consisting of fine-grained quartz and feldspar, were found all through the alaskite, but particularly along the contact. They are evidently slightly later intrusions from the same magma; for they are in some cases found grading into coarse alaskite.

3. *Alaskite-Porphyry.*

Distribution.—The alaskite-porphyry is intimately associated with the ore-deposits, and is best developed in the northern portion of the lime belt, where it forms a great stock bounded on the south, east, and west as shown in Fig. 8, and disappearing beneath the detrital material of the desert to the north. Its extent NE. beyond the Metallic Bobby claim (6) is unknown. Southwest of the main limestone block, it pinches out to a thin streak; but at the SE. end of this block on the Young America property it forms an appreciable mass limited by quartz-porphyry, alaskite, limestone, and the detrital plains of the desert.

The contact between the limestone and the alaskite-porphyry is in some places obscured by a thin layer of float, and by the

puzzling character of the altered rocks, and could, therefore, be only approximately located on the map; but the general relationships are believed to be accurately shown.

Description.—The alaskite-porphyry shows considerable variation, but the generally acidic character of the rock and the lack of dark silicates seem to justify this name, which was first given by C. F. Tolman, Jr.[6] As found in the field it is most often a buff-weathering, fine-grained rock, apparently made up of minute grains of quartz and feldspar (from 1 to 2 mm.) set closely in a dull white ground-mass. The feldspar is milky white, and so dulled by alteration that it offers little contrast to the ground-mass. It is very likely largely orthoclase; and what little plagioclase is present is probably not more basic than oligoclase. The determination of the feldspars, however, is unsatisfactory; for they are so packed with sericite and other alteration-products that extinction and index of refraction are usually undeterminable. The ground-mass is now a mixture of sericite, minute grains of feldspar, and mosaic quartz, the latter largely secondary. It was surely at one time in part glassy, but may have been in part felsitic. Apatite is an important accessory mineral.

Although the above represents the average alaskite-porphyry, several variations are to be noted. The rock forming the high hills on the Jesuit (1) and John F. (9) claims weathers more red than buff, and has a predominance of plagioclase, some of which is andesine. In connection with the origin of the biotite-granite, it is significant that the alaskite-porphyry, in several places, notably in the Emerald claim, contains microscopic biotite.

As all of the alaskite-porphyry is excessively silicified, it is possible that it originally contained, as an essential mineral, biotite, which has since been entirely replaced by quartz. A careful search for traces of this alteration, however, revealed few suggestions of such a change.

Along the contact of the alaskite-porphyry and the limestone are patches of what is known locally as quartzite—a name justified by the appearance of the rock; for it is to the naked eye a quartzite in every respect except that the presence of jointing

[6] *Mining and Scientific Press*, vol. xcix., No. 22, pp. 710 to 712 (Nov. 29, 1909).

suggests an intrusive. Microscopically, it is a mass of quartz grains with interstitial sericite, and undoubtedly represents an extreme silicification of the alaskite-porphyry, Figs. 10 and 11. This conclusion is based upon the study of many thin sections, which showed all stages from a partial replacement of the ground-mass by mosaic quartz, to a complete silicification with obliteration of all original structures of the alaskite-porphyry. The details of this silicification furnished some interesting petrography which cannot be discussed here. The difference between this "quartzite" and the quartz dikes will be discussed later.

Age.—The lack of good contacts between the alaskite and the alaskite-porphyry caused some difficulty in determining the relative age of these two rocks, a point of considerable economic importance. The alaskite-porphyry is considered to be younger, for the following reasons: (1) the exceedingly coarse alaskite shows no change in texture at the contact; (2) in the Hilda claim, the alaskite-porphyry seems to form a dike in the alaskite; (3) the contact dips under the alaskite; (4) it is most in accord with our knowledge of igneous rocks to have a large plutonic mass like the alaskite followed by a supplementary intrusion of smaller extent and porphyritic texture.

4. *Quartz Dikes.*

In the gulch in the Mollie claim there is in the alaskite-porphyry a dike 4 ft. wide with well-defined walls and showing the prismatic jointing characteristic of igneous rocks. The hand-specimen looks exactly like a light gray quartzite. Microscopically, it proves to be made up almost entirely of interlocking quartz grains, very dusty and full of inclusions, some of which suggest zircon by their shape, while others have the needle-like form of rutile, Fig. 12. A few flakes of sericite and some iron-decomposition products are present. Except for the abundance of inclusions in the quartz and for the well-defined field-relationships, there is nothing in this rock to distinguish it from the extreme phases of silicified alaskite-porphyry, and it is probable that other occurrences of this quartz dike have been passed over as silicified alaskite-porphyry. In the Mammoth mine such a dike has been traced in the five upper levels, but is not indicated on the map. This

rock is interesting as furnishing another instance of the gradation between fissures filled with quartz and acid intrusives.[7] It does not, however, show the feldspar phenocrysts found in similar rocks in the Yukon gold-district.[8] It is found only in the alaskite-porphyry, and is not much younger than that rock.

It might be urged that all the rock referred to above as silicified alaskite-porphyry is actually quartz dike. I believe, however, that the so-called "quartzites" of the district are in some cases dike-rocks and in others alterations of alaskite-porphyry. That some are dike-rocks is plain from the occurrence just described; that some are silicified porphyries seems equally plain from the transitions shown by thin sections, in which all stages have been traced, from partial replacement of the ground-mass by quartz to a complete silicification in which the former presence of feldspar phenocrysts is indicated only by the shape and kaolin-like inclusions of some of the quartz masses. While it is justifiable to believe that these quartzites are sometimes of one origin, sometimes of another, it is not always possible to class an isolated outcrop; and all have been mapped as a part of the alaskite-porphyry.

5. *Biotite-Granite.*

Under this heading are included a true granitic rock, and a porphyry which represents contact- and dike-facies of the same magma.

Distribution.—The biotite-granite proper forms two irregular masses on the claims of the Imperial Copper Co. (Fig. 1), while the porphyritic phase occurs as dikes in the limestone and the alaskite-porphyry. Many of these smaller masses are not shown on the map; but they are most abundant in the vicinity of the larger areas. Their peculiar shape will be discussed below. Biotite-granite is not as prominent on the Young America property as on that of the Imperial Copper Co.

This rock weathers to a buff color similar to that of the alaskite-porphyry, and when fine grained is to be distinguished from the latter only by the presence of biotite. Silicification may destroy this mineral; and then differentiation of the two

[7] Spurr, *Professional Paper No.* 55, *U. S. Geological Survey*, p. 129 (1906); and Lawson, *Bulletin*, Department Geology, University of California, vol. iii., No. 17, p. 388 (1902–04).

[8] Spurr, *Trans.*, xxxiii., 310 (1902).

rocks is possible only by microscopic search for remnants of mica. In view of these conditions, the outlines of biotite-granite, as mapped, must be considered as expressing only approximate relations.

Description.—The biotite-granite forming the large masses is a holocrystalline rock of quartz, feldspar, and biotite, weathering readily to a buff friable rock, in which the hexagonal plates of greenish biotite are very prominent.

Only the porphyritic phases found as smaller masses and as dikes were compact enough for thin sections. These have a light gray to white ground-mass with large (0.5 cm.) phenocrysts of biotite and pinkish feldspar, and smaller crystals of white feldspar and quartz. The ground-mass, when not altered, is in some cases a close intergrowth of feldspar and quartz (0.005 mm.), while in other specimens traces of glass were found. The feldspar is more abundant than the quartz, and the relative amounts of orthoclase and plagioclase vary, the former usually predominating. The lime-soda feldspar is in the oligoclase-andesine series, and often weathers to epidote. The biotite was evidently titaniferous, for leucoxene is always among its alteration-products. Sphene and apatite are very important accessories; and some perofskite was noted.

Origin.—The magmatic relation of this biotite-granite to the other intrusions is of great importance in connection with the origin of the ores. In this connection, three points are significant: (1), the limitation of the rock to areas of alaskite-porphyry; (2), the mineralogical similarity between the alaskite-porphyry and the biotite-granite, especially as the former does, in some cases, contain black mica; and, (3), the peculiar shape and contact-relations of the biotite-granite. In regard to this, it should be said that the biotite-granite often occurs in the alaskite-porphyry as bunches, lenses, and irregular masses, which finger out into the wall-rock, and are mixed with it in a way not to be explained by ordinary intrusions along fissures. Although in some cases it shows signs of finer grain, due to chilling along the contact, small masses of it may be seen holding their holocrystalline texture throughout. Very irregular lenses are often found, that suggest inclusions caught up by the alaskite-porphyry rather than ordinary intrusions. In fact, when first seen in the field, the irregular masses of

granite were explained in this way; and this hypothesis was abandoned only when well-defined dikes of granite-porphyry were found in the alaskite. In view of the above facts, it is suggested that this rock represents a somewhat basic differentiation of the magma which supplied the alaskite and alaskite-porphyry, and that it was intruded shortly after the latter, which had not completely solidified. Where this biotite-bearing differentiation-product found the alaskite-porphyry still warm and pasty, it formed irregular tongues of biotite-granite; where the alaskite-porphyry had already solidified and fissured (on the contacts), it formed dikes of biotite-granite-porphyry.

The possible objection to the above theory is the well-known fact that in the splitting of a magma, the acidic intrusions more often follow than precede the basic.[9] On the other hand, it has been suggested that a differentiation of a rock-magma may take place through the crystallization and sinking of some of the mineral constituents.[10] Through this process biotite may have been concentrated in the depths of the parent magma of the Silverbell rocks; the upper (mica-free) portions being intruded or "tapped off" first, and the lower (biotitic) portions afterwards squeezed up by some later crustal movement. All discussion of magmatic differentiation is largely speculative; but the above theory explains so many features of the local geology, that it seems justifiable to consider Silverbell as a district where rock-differentiation by fractional crystallization is a very useful working-hypothesis.

6. *Quartz-Porphyry.*

Distribution.—This rock covers the entire eastern part of the area mapped, its boundaries being shown in Fig. 8. In general, it forms the higher hills to the east, and for this reason, and because of the large amount of glassy ground-mass, it may be taken for a surface-flow. If this is true, the portion remaining at the present day must represent the bottom of a very thick flow, for it is entirely free from gas-cavities. Moreover, its contact with the alaskite-porphyry, where exposed in the gulches, is vertical, and it has been found sending dikes into

[9] Spurr, Igneous Rocks as Related to Occurrence of Ores, *Trans.*, xxxiii., 288 to 340 (1902). See also Harker, *Natural History of Igneous Rocks*, chap. V.

[10] Harker, *loc. cit.*, and Iddings, *Igneous Rocks*, p. 251 *et seq.* (1909).

the limestone of the Silverbell (51) and Black Daisy (54) claims, and into the alaskite-porphyry of the gulch in Northern Star (15). I am inclined to regard it as an intrusive which came in considerably later than the alaskite series, after erosion had removed much of the overlying rock, so that it solidified under smaller pressure.

Description.—In the hand-specimen, the quartz-porphyry shows a dense black ground-mass, thickly studded with white feldspars about $\frac{1}{16}$ in. (rarely 0.25 in.) in diameter. It also contains angular black fragments, sometimes irregular in shape, at other times roughly rectangular, and bent as if by flowage. These probably represent fragments of glass from a first cooling, broken up by a partial remelting after the first solidification.

Weathering so bleaches the ground-mass that outcrops of the quartz-porphyry are light gray, offering less contrast in color to the alaskite-porphyry than one would expect from the appearance of fresh specimens. This gray is, however, distinctly different from the buff of the latter rock, and this, together with the presence of glassy fragments and many knots and veinlets of epidote in the quartz-porphyry, makes field-separation of weathered material comparatively easy—a much appreciated condition in this region of rocks of confusing similarity.

Under the microscope we see a true glassy ground-mass which owes its black color to an abundance of microlites and trichites. In this are scattered the phenocrysts of quartz, feldspar, and biotite. The extensive resorption of the first named is notable. The feldspars seldom show polysynthetic twinning, but in many cases have an index of refraction greater than balsam, so that much of the feldspar must be plagioclase, somewhere below oligoclase in the series. Foils of biotite varying in length from 0.1 to 1.0 mm. are common.

In view of the importance of the soda-lime feldspars as seen in thin sections, it is possible that this rock is more closely allied to the dacites than to the rhyolites. It is, however, possible that the glassy ground-mass represents a large amount of uncrystallized orthoclase; and for this reason, and because of the great abundance of quartz, I prefer to use the term quartz-porphyry, with the reservation that a chemical analysis might prove the rock to be a dacite. "Quartz-porphyry" is used instead of "rhyolite-porphyry" because the latter has been ap-

plied by mining-men to the alaskite-porphyry, and also because "quartz-porphyry" is a somewhat more elastic term.

This rock is little altered when compared with the alaskite intrusives, there being neither extensive silicification nor sericitization. The usual products of weathering by surface-waters are, of course, present, and the inclusions of glass are often changed to epidote. It is notable that the mica does not decompose to the leucoxene masses found in the biotite-granite, and is probably not titaniferous.

That the quartz-porphyry is definitely younger than the alaskite-porphyry, and that it has not undergone the extensive mineralization seen in the latter, are facts of considerable economic importance.

7. *Andesite.*

The most conspicuous dike-rock in this district is an andesite of later age than the ores, and of no economic importance. It occurs most frequently along a NW–SE. line from the Emerald- (49) Prospector (48) boundary to the railroad bend in the Pope (2) claim. In general, there seem to be three dikes, one varying from 15 to 30 ft., while the other two are only from 2 to 4 ft. in width. The largest is the "Birdseye" dike of the Mammoth mine. All dip from 80° to 85° to the SW. The most significant thing about these rocks is their irregular distribution. They are not the result of filling a straight and well-defined fissure, but form "corrugated dikes," showing great variation in width. They outcrop only occasionally at the surface and even then show as irregular patches, with greatly varying strike. The present topography is evidently at about the upper limit of the intrusion, for exposures in gulches show andesite bounded on the top and sides by alaskite-porphyry. It was only by locating all outcrops and by aid of the underground maps that their true relations were determined.

In appearance the andesite is a dark-gray, fine-grained rock, sometimes showing mica and feldspar phenocrysts. The microscope shows a mass of feldspar laths, largely oligoclase with occasional orthoclase, and flakes of biotite, sometimes bleached by weathering. Somewhat frequently quartz was found between the feldspar laths; but this rock is distinctly more basic than the quartz-porphyry.

None of this andesite was found in the quartz-porphyry and I am inclined to consider the latter as the older of the two.

8. *Miscellaneous Dikes.*

East of the quartz-porphyry, outside of the area mapped (Fig. 8), there is a complicated series of intrusive rocks and surface-flows entirely different from any rocks thus far described; and within the property of the Imperial Copper Co. were found some dikes which may be related to them, but which are locally unimportant, and will be mentioned only briefly.

Rhyolite-Porphyry.—This forms, in the eastern end of the Long Shot (52) claim, a 30-ft. dike, which circles around to the north, just off the limits of the large map. It also occurs in the vicinity of the lead-silver prospects to the east (Fig. 8). It is more resistant than the quartz-porphyry in which it occurs, and stands out as a sharp wall from 10 to 20 ft. high. It has a dense pink ground-mass, with phenocrysts of quartz (2 mm.), orthoclase (3 to 4 mm.), and a few smaller biotites. The ground-mass is finely micro-crystalline, and zircon and magnetite are important accessories.

Hornblende-Andesite-Porphyry.—Small outcrops, not showing the whole thickness, were found in the Confidence claim, and much float of this material is present in the vicinity. The rock has a dark-green ground-mass, thickly studded with phenocrysts of plagioclase, in the labradoritic series. The ground-mass is a mixture of microscopic hornblende, a little biotite, and feldspar. The last is probably largely orthoclase, for it has an index of refraction less than balsam, and shows no polysynthetic twinning. In view of the uncertainty of the amounts of alkalic and calcic feldspars, classification is difficult; but the large amount of hornblende and the basic character of the abundant phenocrysts make it more closely allied to the andesites than the trachytes.

A tunnel in the John F. claim has exposed a similar dike, in which biotite largely replaces the hornblende.

IV. Age of the Rocks and Comparison with Other Localities.

It has already been noted that the limestone at Silverbell is probably Carboniferous, thus making the intrusive rock post-

Palæozoic in age. Prof. C. F. Tolman, Jr., has pointed out the general association of the copper-ores of southwestern Arizona with granitic rocks Mesozoic or younger in age.[11] Table I., which shows the kind and age of the post-Palæozoic intrusions of neighboring copper-districts, is therefore of interest. The names of rocks supposed to be genetically connected with the ores are italicized.

TABLE I.—*Post-Palæozoic Intrusives in Several Copper-Districts of Arizona and New Mexico.*

	Globe.	Bisbee.	Morenci.	Miami.	Ray.	New Mexico in General.
Tertiary.	Dacite.		Rhyolite, basalt and andesite lavas.			Rhyolite, basalt and andesite lavas.
	Schultze granite[a]		Diabase. *Granite-* and *Monzonite-porphyries.*	*Schultze granite*[a]	Dacite. *Granite.*	*Monzonites.*
Mesozoic.	Diabase. Monzonite.	*Granite-porphyry.*				

[a] Schultze granite is probably Tertiary, according to Weed and Tolman; *Mining and Scientific Press*, vol. xcix., p. 356, 1909. Ransome considers it probably Mesozoic; *Ibid.*, Feb. 12, 1910.

Exact determination of the age of an intrusive in any of the districts mentioned in this table may be uncertain, because the igneous rocks are not always in decipherable relations to sediments of known age. It is at once apparent, however, that the copper-ore deposits in Arizona and New Mexico are closely connected with a series of acidic intrusions which took place either at the close of the Mesozoic or at the beginning of the Tertiary, and that these acidic rocks were followed by less important intrusions of somewhat more basic character. This generalization applies very well to Silverbell, although in that district the greater part of the earlier intrusives is so free from dark silicates as to justify the name of alaskite, whereas the granites and monzonites of the other districts contain ferromagnesian minerals. This only strengthens the hypothesis advanced, that the normal magma from which these intrusions came was granitic, and that the alaskites are due to local dif-

[11] *Mining and Scientific Press*, vol. xcix., No. 11, pp. 356 to 358 (Sept. 11, 1909); No. 12, pp. 390 to 392 (Sept. 18, 1909).

ferentiation. It is also true that there is a great variation in the nature of the feldspars of these alaskites; the predominance of plagioclase in some specimens showing a transition to granodiorite. At Globe, Morenci, and Ray the granitic intrusions have been followed by more basic rocks—diabases or dacites, and the same general succession holds true for Silverbell, where andesite and a basic quartz-porphyry (dacite?) followed the alaskites. The complex series of later rocks occurring east of the area mapped will probably be found to correspond very closely to the Tertiary flows observed by Lindgren, Graton, and Gordon in New Mexico,[12] and by Lindgren at Morenci.

V. Alteration.

Under this head will be discussed all changes not definitely connected with contact-metamorphism or the actual deposition of ores, both of which will be treated in separate chapters. Apart from the disintegration of the granite, the effects of weathering are not extensive. The limestone and hornfels are coated with black "desert varnish," due to the deposition of manganese and iron oxides by the evaporation of vadose water;[13] joints of the igneous rock show dendritic coatings of secondary manganese oxide; and the usual alterations of feldspar to sericite, kaolinite, calcite, and occasional epidote, and of biotite to chlorite, are seen in thin sections. The buff and red staining of the outcrops of alaskite and granite is due to oxidation of disseminated pyrite.

More extensive and of greater interest is the alteration due to hot waters, probably of magmatic origin. This is of two kinds, sericitization and silicification; and its age and cause are fairly well established.

1. *Sericitization.*

Specimens of both granite-porphyry and alaskite-porphyry, even when taken from below the water-level, and apparently perfectly fresh, with bright pyrite cubes scattered through them, invariably show in thin section a ground-mass packed with sericite and quartz. The sericite also fills a large part of

[12] *Professional Paper No.* 68, *U. S. Geological Survey* (1910).
[13] Blake, W. P., *Trans.*, xxxv., 371 to 375 (1904).

the feldspar phenocrysts, often making the identification of the kind of plagioclase impossible. Microscopic veinlets of sericite are seen in nearly all sections (Fig. 13), and it is because these veinlets are often cut by quartz veinlets that the relative age of the two processes is known. Van Hise, quoting Lindgren, considers sericitization as the result of the action of carbonated waters upon feldspar. It may therefore be due to surface-waters; but if this had been the case at Silverbell, the andesites, quartz-porphyries and miscellaneous dikes should show similar effects. Although they do show some sericite—the result of surface-weathering—it is entirely distinct from the widespread sericite in the alaskite-porphyry and granite.

2. *Silicification.*

This is a striking feature of the alaskite-porphyry and granite-porphyry, and shows clearly in hand-specimens. In many places these rocks are cut by veinlets of quartz, or show lens-like inclusions of secondary quartz. In some instances quartz veins and lenses several feet in width are found, but masses an inch or less in diameter are more common. The alaskite-porphyry forming the hill-tops in the Jesuit, John F., Ruby, and Billy claims is much silicified, the replacement of the ground-mass and of the feldspars by quartz being clearly visible to the naked eye. In fact, the superior resistance of this siliceous rock seems to have been a factor in determining elevations. Microscopic examination of the alaskite-porphyry and the biotite-granite-porphyry shows that all of the rock has been attacked, to varying degrees, by siliceous waters. Making due allowance for the SiO_2 set free in the process of sericitization, it is evident that much quartz has been brought in. All stages have been observed, from that in which a few patches of mosaic quartz are found in the ground-mass to a complete change of the ground-mass to quartz and a little sericite. In some cases the quartz still preserves traces of the feldspar form and structure. Often microscopic veinlets traverse the rock, and silicification has taken place from these. In some slides, veinlets of mosaic quartz were found cutting through original quartz phenocrysts; in others, the single crystals of quartz had been changed to a mosaic of small grains all differently oriented (Figs. 10 and 11).

As a rule, the silicification is strongest along the lime-contact, where it often produces "quartzite." As will be shown in the discussion of contact-metamorphism, silica-bearing waters evidently attacked the limestone itself. The existence of the quartzose cappings on the hills, mentioned above, is somewhat difficult to explain. It may be that the limestone was formerly more extensive, and that these hill-tops were once alaskite-porphyry, which formed a contact with limestone.

The quartz-porphyry and the later dikes are closely associated with the alaskite-porphyry and the granite, and there is nothing in their composition or structure to render them less susceptible to the action of mineralizing solutions; yet they do not show the changes just described. The sericitization and silicification, therefore, are attributed to magmatic waters rising along the granite and alaskite-porphyry shortly after their intrusion, and before that of the quartz-porphyry. That these waters did not migrate far from the rocks with which they were associated is shown by their slight effect on the coarse alaskite.

VI. Structures.

1. *Dip of the Limestone.*

Many attempts were made to determine the dip and strike of the limestone, but it is now altered to so massive a marble that former bedding-planes are not recognizable with any certainty. By considering the siliceous streaks and the hornfels as marking former beds, the general conclusion was reached that the strike varies from N–S. to NE–SW., while the dip is W. or NW., and nearly vertical. Careful observations taken at separate points, however, show such wide variation, and the discontinuous character of the hornfels beds is so noticeable in the underground workings, that we are forced to believe that great heat and pressure at one time changed the limestone to a mass of wax-like consistency, which the thrusts of successive intrusions molded and "kneaded" until the original structures were destroyed, and former beds were no longer continuous.

2. *Faulting.*

This district presents at first sight such an abundance of shearing-planes, shattered rock, and slickensided surfaces, that

important faulting is expected. A careful plotting of the dip and strike of these apparent lines of movement, both on the surface and underground, shows, however, that they have no uniformity, and seldom, if ever, connect to form well-defined lines of dislocation. They seem to be due to very small rock-movements, resulting either from the forces of the many intrusions, or from the readjustments attendant upon frequent heating and cooling. That there are true faults in the district is certain; but these involve such small movements, and the rock-contacts are themselves so irregular, that their detection is difficult. Time and again slip-planes were carefully traced in the mines until they met dikes or contact-lines, without finding any displacement. Moreover, the experience of the engineers has been that the ore-bodies have not been seriously disturbed by rock-movements. The faults that have been recognized fall into two classes, of different strike and age.

The earliest faulting took place after the intrusion of the andesite dikes, and followed them in strike and dip. It is recognized chiefly by the strong shearing and slickensides which are seen along the hanging-wall of the large dike in the Mammoth mine, where it is known as the "Mother Fault." Slickensides also appear on the wall of an andesite dike in a tunnel running from the Southern Beauty (24) to the Union (29) claim, and the andesite dikes often show sheared surfaces; but in no case has appreciable displacement been recognized.

Lines of NE–SW. faulting later than the above were found as shown on the map, Fig. 8. The most northerly of these faults was detected by tracing brecciated and slickensided rock to the lead-silver claims at the northeast of the district, where a rhyolite dike shows a horizontal displacement of about 150 feet.

Just east of the limits of the large map, in the gulch running down the Northern claim (30), the displacement of a rhyolite dike shows a fault, the rocks on the northern side having moved west about 50 ft. Faulting has not taken place along a single plane, but along many slickensided surfaces with considerable variation in strike, though the general line of movement has been NE–SW. What is probably the continuation of this fault is shown in the tunnel on the Southern Beauty (24) and Union (29) claims, where an andesite dike is displaced. The movement here, however, is not over 3 ft., indicating, appa-

rently, that this fault, which showed in one place a 50-ft. displacement, distributed its movement, farther on, among several widely-spaced planes. The similarity in shape of the two masses of biotite-granite in this vicinity suggests that one has been faulted down from the other, and a continuation of the above-mentioned fault-plane down the gulch in the Hilda claim (26) would fall in the right place for such a dislocation; but the small displacement shown by the dikes argues against this, and I am not inclined to believe that any such extensive vertical movement has taken place.

The third fault, shown in Fig. 8, on the Young America property, was detected by the presence of fault-breccia and a displacement of the contact between alaskite and alaskite-porphyry.

VII. Contact-Metamorphism.

The changes in texture and in mineral and chemical composition which have taken place at the contacts of intrusive rocks are not only theoretically important, but also closely connected with some of the ore-deposits; and an understanding of them is necessary before the genesis of the ores can be discussed. The contact-metamorphism is confined to the contacts of the limestone with the alaskite-porphyry and the biotite-granite-porphyry. It will be discussed under two heads: (1) the changes within the intrusives (endomorphism); and (2) the changes in the limestone (exomorphism).

1. *Endomorphism.*

It has long been noted that, except the textural differences due to quicker cooling, and occasional segregations on the margins, intrusives are seldom extensively changed at the contacts with wall-rocks, and this generalization holds true in the Silverbell district.

The most notable change, the silicification of the alaskite-porphyry and the biotite-granite porphyry on the limestone-contacts, is not an effect of the sediment, but is the result of siliceous waters rising from the depths of the magma and percolating slowly through the solidified margins of the intrusives.

It is reasonable to suppose that if assimilation of intruded sediments and consequent change in composition of the intrusive were a common occurrence, it would be well shown at

Silverbell, where the limestone was literally swamped in the igneous rocks, and the excess of molten material must have furnished more heat than is present when a sedimentary series is cut by igneous rocks of smaller extent. The fact is, however, that the contacts in this district very rarely show an increase in the lime-content of the intrusive.

In all cases, the boundary between limestone and alaskite- or granite-porphyry is sharp, and in a few instances only do the igneous rocks show an increase in lime-minerals. Of 20 or more thin sections of igneous rocks from the lime-contact, only one contained a few crystals of epidote, which, from their shape and included position in quartz, seemed to be pyrogenetic. One other slide showed hornblende, which might be attributed to the addition of enough calcium to the magma to cause the crystallization of an amphibole in place of the biotite.

In an open pit near the bend of the road in the Hamilton (35) claim there is a peculiar greenish-yellow rock at the contact of alaskite-porphyry and limestone. It is decomposed to a soft, clay-like mass, which defied determination in the hand-specimen; but a thin section showed that it is alaskite-porphyry now largely changed to diopside, which occurs in masses made up of club-shaped crystals from 0.1 to 0.05 mm. in diameter, with which are found occasional garnets. Another rock from the same pit appears to the naked eye like typical alaskite-porphyry, although somewhat silicified and copper-stained. Microscopically, it shows remnants of a felsitic ground-mass of quartz and feldspar, characteristic of alaskite-porphyry, but it is largely made up of an intergrowth of epidote and garnet, the existence of which was never suspected from the hand-specimen. The relationship of the diopside, garnet, and epidote to the rest of the rock in these two specimens is striking, for these minerals give every indication of replacing the original constituents (Fig. 14). The intrusive, therefore, does not appear to have fused into itself some of the limestone, thus forming a rich calcium magma from which epidote, garnet, and diopside crystallized. It seems rather as if the hot magmatic waters which circulated freely in this district after the intrusion were occasionally highly calcareous because of local leaching of limestone fragments, and were the cause of this

formation of lime silicates in the porphyry. It is believed that this calcium-content of these waters was due to leaching of limestone, and was not an original constituent of the magmatic solutions, because the formation of diopside, epidote, etc., in the porphyry is of limited extent and only a local effect, whereas the silicification and sericitization attributed to these same waters is widespread. In this respect, we find a difference from the Velardeña district, Mexico, where magmatic waters rich in calcium and silica changed a great part of the intrusive to a rock of garnet, epidote, and pyroxene much like the typical metamorphosed limestone.[14]

2. *Exomorphism.*

General.—The chief changes in the limestone seem to have been caused by the alaskite-porphyry and the biotite-granite. Little metamorphism is noticeable on the contact with the quartz-porphyry. In some parts of the district not studied in detail (Fig. 8), the limestone is in direct contact with coarse alaskite, and shows changes in no way different from those effected by the alaskite-porphyry or the biotite-granite. But because the extent and intensity of the alteration of the limestone varies directly with the amount of the last-named rocks present, they are believed to be the chief agents in the contact-metamorphism. The granite and the alaskite-porphyry seem to have produced exactly the same results; and no effort will be made to distinguish between effects of the two. The changes produced in the sediments by these two rocks are (1) recrystallization, (2) destruction of color, and (3) formation of irregular masses and bunches of garnet, with subordinate wollastonite, diopside, and other minerals.

Recrystallization.—This is extensive, none of the original clastic structures having been preserved; and although, in conformity with local usage, the terms "limestone" and "lime belt" have been used in this paper, there is no true limestone in the district; all calcareous material is now marble.

The dense siliceous rocks supposed to represent former clayey beds are also completely microcrystalline.

Loss of Color.—It is possible that the original limestones were black or nearly black, and that the white color of much

[14] Spurr and Garrey, *Economic Geology*, vol. iii., No. 8, pp. 688 to 725 (Dec., 1908).

of the marble is due to the driving off of carbonaceous coloring-matter. This view is supported by the finding of some blocks of black marble, which may represent fragments in which metamorphism has not gone very far. Moreover, some of the white marble is streaked with gray, and in some cases the gray predominates, forming a gradation to black marble banded with white. Exactly how much of the original rock was black and carbonaceous it is impossible to tell. A similar change in color has been noted in carboniferous limestone near Tucson.[15]

Formation of New Minerals.—At many places along the contact the limestone has been changed to great masses of garnet, with varying amounts of quartz, wollastonite, and diopside. This garnet rock is very resistant, and is stained black on the weathered surface by manganese, and thus forms the great black caps with which the ore is associated. Garnet in minor amounts may be found anywhere in the lime belt, but actual garnet rock occurs only along the contacts, and is best developed on the northwestern end of the largest "lime block," and in the outliers near by. Its distribution is shown on the large map, Fig. 1, but it always grades into the non-garnetized marble. It is noteworthy that the richest development of contact-metamorphic minerals (and also of ore) is near the largest amount of biotite-granite. Biotite-granite is less important in the Young America property, at the southeastern end of the lime belt, and contact-metamorphic minerals and ore are less abundant there than on the claims of the Imperial Co. Contact-metamorphism, however, is so irregular in this district that no definite conclusion can be based on this fact; but it does suggest that while the alaskite-porphyry was a metamorphosing agent, the granite was more important.

Going from the boundary of the intrusive rock into the sediment, we first pass a band, 3 to 10 ft. thick, of solid garnet and quartz, with varying amounts of wollastonite, and microscopic diopside. This is gradually replaced by garnet rock with veins and bunches of marble, and the latter becomes more abundant until we find marble with lenses and stringers of garnet and associated minerals, and these become less numer-

[15] Blake, W. P., Report of Governor of Arizona for year ended June 30, 1904, p. 72.

ous until a pure marble remains. There are so many dikes in this district, and the contacts are so irregular, that it is impossible to say how far from the intrusive mineralization has extended, but 500 ft. is probably the maximum. The relation of the garnet to the marble is remarkable. It apparently follows no former bedding-planes or lines of fissuring, but forms stringers, short veins, and lenses, strongly suggesting the occurrence of pegmatite masses in some gneisses. An open pit northwest of the Mammoth shaft in the Hamilton claim (35) gives a good section of the solid garnet rock on the contact. Here two types can be recognized: a wollastonite rock, composed largely of dense white wollastonite with subordinate garnet, and a garnet-quartz rock in which rich brown garnet and colorless quartz are intermingled in masses of varying size. The second rock cuts through and includes fragments of the first.

The garnet rock shows microscopically the same characters whether taken from solid masses on the contact or from lenses farther away from the intrusive. There is considerable variation in the relative amounts of the minerals, but otherwise the following description will apply to the typical garnetized marble. The rock is an intricate mixture of garnet, quartz, wollastonite, diopside, and calcite. Wollastonite is invariably the first mineral formed, and diopside (which is comparatively rare) is next in age (Fig. 15). The quartz and garnet are intergrown in a puzzling way, apparently having been formed at about the same time. Some rocks show a ground-mass of finely crystalline calcite, which may represent the residue of the original limestone; but a careful search through many sections failed to reveal any definite evidences of garnet replacing original calcite. All the rock-specimens show veinlets of quartz and calcite later than the previously-mentioned minerals. In many slides it seemed as if garnet had formed with its own crystal boundaries, leaving cavities into which quartz and calcite solutions had afterwards filtered. (Figs. 16 and 17.)

The lack of epidote is remarkable, for in other districts it very often accompanies garnet as a contact-metamorphic mineral. It has been found in a few places, close to the contact, but is usually rare in the altered limestone. Some lenses of quartz were seen in the marble, surrounded by reaction-rims of

wollastonite, similar to those described by Professor Crosby.[16] The contact-metamorphic minerals in this district are comparatively few, those recognized being garnet, quartz, wollastonite, diopside, epidote, scapolite (?), and plagioclase. It is to be noted that minerals with fluorine, boron, and other so-called "mineralizers" are practically absent—which coincides with the description of most other contact-metamorphic ores.

The Origin of the New Minerals.—In a study of contact-metamorphism four problems always arise, viz.: (1) What, if any, substances did the intrusive add to the sediment; (2) what changes in volume have taken place; (3) what determined the loci of metamorphism; and (4), how were the new minerals formed? The most suggestive discussions of these questions have come from Professors Kemp, Barrell, Vogt, Stutzer, and Leith, and Messrs. Lindgren and Weed. References to important papers will be found in the appended bibliography.

Addition of Material.—In determining whether contact-metamorphic minerals are the result of recrystallization of impurities originally present in the limestone, or of substances added by waters given off by the intrusive, a knowledge of the composition of the unaltered limestone is essential. In the Silverbell district the destruction of all bedding makes it impossible to follow a definite garnetized layer into its unaltered equivalent. Moreover, there is no absolutely unmetamorphosed limestone in the area studied. In the marble farthest away from the contact there are lenses and stringers of more siliceous material, the shape and position of which so correspond to the lenses of garnet, that it might be thought that recrystallization of similar masses of impure limestone gave rise to the characteristic minerals of the garnetized belt. An analysis of one of these lenses of impure marble gave the following:

SiO_2	17.46
Al_2O_3, Fe_2O_3	1.75
CaO	45.76
MgO	2.26
Ignition-loss	33.31
	100.54

[16] Crosby, W. O., Limestone-Granite Contact-Deposits of Washington Camp, Arizona, *Trans.*, xxxvi., 626 to 646 (1905).

This rock is crystalline, although much finer-grained than the average marble; and that it contains some wollastonite is shown by the microscope, and by recalculation, considering the loss on ignition as CO_2, so that it does not represent unaltered sediment. It is notable that the chief impurity is SiO_2, and that Fe_2O_3 and Al_2O_3 are very low. An analysis of the purest and freshest garnet obtainable gave the following:

SiO_2	36.85
Al_2O_3	9.62
Fe_2O_3	17.54
FeO	1.62
MgO	1.11
CaO	31.87
MnO	0.51
H_2O—110°	0.27
	99.39

Some CO_2 was present, but was not determined. A recalculation shows that the garnet is a mixture of the following species in the proportions given:

Andradite,	53.4
Grossularite,	37.4
Pyrope,	4.4
Almandine,	3.5
Spessartite,	1.2

A little silica and calcium are left, which are accounted for by the quartz and calcite found in thin section.

This agrees with the work of Professor Kemp and Mr. Lindgren, who have shown that the contact-metamorphic garnets are usually predominantly of the lime-iron variety. In view of the fact that the most impure lime rock in this district contains such insignificant amounts of iron and aluminum, the inevitable conclusion is that here, as in so many other districts, the garnet has been formed by reaction between the calcium of the original sediment, and silica, iron, and alumina brought in by hot waters given off by the intrusive. This idea is confirmed by finding with the garnet the great masses of quartz, which in size and field-relationships show no sign of being metamorphosed sediments. It is possible, on the other hand, that the wollastonite is, in part, at least, the result of the recrystallization of siliceous portions of the limestone. This

view is strengthened by the observed reaction-rims between marble and flint nodules. Both Professor Barrell and Mr. Weed have shown that wollastonite is often formed by recrystallization, even in districts where additions from the intrusive characterize the contact-metamorphism. In such cases the wollastonite formation takes place first, and the resulting porosity makes room for the entrance of solutions from the magma. The fact that at Silverbell the wollastonite is the first formed of the contact-minerals suggests that a similar sequence of events has taken place there.

It is not necessary to assume addition of material to account for the minerals in the hornfels; they may be due to a recrystallization of a siliceous shale.

Volume-Changes.—When an impure limestone is recrystallized to a mass of contact-metamorphic minerals, a notable shrinkage in volume must take place. Into the pores thus formed may come magmatic waters bearing silicate solutions and ores.[17] Recrystallization, however, has not played an important part at Silverbell, yet there is evidence of the formation of considerable open space. Microscopically, the garnet often shows crystal boundaries, and a characteristic of many thin sections is garnet with cavities and veinlets filled with calcite and quartz. Small vugs lined with garnet are sometimes seen with the naked eye, although, as a rule, the garnet rock appears very dense, because all cavities have been filled by later minerals. Moreover, the ores are distinctly later than the garnet, in which they occur as veinlets or as disseminations. In accounting for the formation of these cavities we must take into consideration the following facts. The formation of andradite by the addition to a limestone of iron and silica, and the removal of CO_2, means an increase in volume. If the original limestone had a porosity of 10 per cent. and the resulting andradite were practically non-porous, the total increase would be roughly 20 per cent. On the other hand, the change from a porous limestone to a dense marble involves a shrinkage, probably of 10 per cent. in the case of a limestone of average porosity. Now, the greater part of the limestone at Silverbell has been recrystallized to marble, and only a minor amount, even near the contact, has been changed to garnet.

[17] Weed, *Trans.*, xxxiii., 715 to 746 (1902).

If we assume that one part of limestone has been garnetized with an increase of 20 per cent. in volume, and nine parts have been marmorized with a decrease of 10 per cent., the net shrinkage resulting from the two processes will be 7 per cent. of the original volume. The fact that this shrinkage caused porosity in the garnet rock, and not in the marble, may be accounted for by magmatic waters under great pressure being abundant near the contact and thus keeping the pores open.[18] These waters may also have dissolved out considerable calcite and redeposited it away from the contact. The well-defined veinlets may be due to contraction on cooling. Whatever may be the cause, the fact remains that the garnet rock at Silverbell was characterized by microscopic pores as well as by well-defined minute fissures, which were afterwards filled by other minerals. It does not seem probable, however, that these open spaces were extensive enough to account for all of the ore-deposition, some of which was very likely due to replacement of calcite and garnet.

Locus of Metamorphism.—Since, in this district, contact-metamorphism and ore-deposition have gone hand in hand, the determination of the factors causing metamorphism in some parts of the limestone and not in others would be of great practical value. In other regions, the following reasons have been assigned for the irregular distribution of the contact-effects. (References will be found in the bibliography.) The more ready access afforded the solutions by bedding-planes or fracturing of the sediments is supposed to have been a determining factor at Bingham, Cochise, and Taylor Peak. At Marysville, Mont., and Sonora, Mexico, the porosity resulting from recrystallization of impure limestone has determined the place of entrance of magmatic waters, while at Washington Camp, Ariz., metamorphism is limited to impure beds, although here no addition of silicates from the intrusive is considered probable. At Morenci the purity and porosity of the limestone were the favorable conditions for contact-metamorphism, while local irregularities were accounted for by variations in the water-content of the intrusive. At Silverbell there is the possibility that the changing of patches of siliceous limestone to

[18] In regard to cavity-formation by water under pressure, see L. C. Graton, *Bulletin No.* 293, *U. S. Geological Survey*, pp. 59 to 60 (1906).

wollastonite formed porous rock, which admitted the magmatic emanations more readily; but in view of the subordinate importance of the wollastonite this is not probable. The two more effective factors were fissuring in the limestone and local variation in the amounts of water given off by the intrusive. It is not to be understood that contact-metamorphism along well-defined fissure-planes has been recognized. The complete "stewing up" of the limestone prevents this; and it is simply suggested that lines of fissuring in the original rock allowed the magmatic emanations their first entrance. The complete obliteration of all original structures prevents any conclusion of practical importance.

Method of Precipitation.—It has long been asserted as a matter of course that contact-metamorphic minerals were formed by siliceous solutions from the intrusive, reacting with the calcite of the limestone, and replacing it metasomatically, molecule by molecule. Professor Stutzer [19] has suggested that it is possible for the magmatic waters to dissolve the calcite, thus forming a solution consisting of the iron, silica, etc., from the intrusive plus the lime from the sediment; that this solution may eat its way into the limestone until saturation and cooling cause it to precipitate contact-metamorphic minerals, much as minerals are precipitated in pegmatite veins. This is a somewhat conjectural question; yet I cannot but think that the lack of microscopic evidence of replacement, the sharp boundaries of the garnet stringers and lenses, and their peculiar field-relations, suggesting "eating into" the marble, are best explained, not as the result of molecular replacement, but as precipitation from highly-heated solutions, which worked their way into the sediment under great pressure, much as pegmatitic stringers have formed in the injection-gneisses. Under this hypothesis, the small pores in the garnet rock would be similar to miarolitic cavities in pegmatites.

VIII. Ore-Deposits.

The ore-deposits of the Silverbell district are of three kinds: contact-metamorphic deposits of copper; copper-ore disseminated in igneous rocks; and lead-silver veins.

[19] Stutzer, O., Kontaktmetamorphe Erzlagerstätten. Zusammenstellungen und Betrachtungen. *Zeitschrift für praktische Geologie*, vol. xvii., pp. 145 to 155 (Apr., 1909).

1. *Contact-Metamorphic Copper-Ores.*

General.—The ores of this class, which have furnished practically all the product of this district, have been worked in three mines, all owned by the Imperial Copper Co.: the Mammoth, the Union, and the Billy; the last two being close together and connected underground. The Mammoth has furnished the richest ore, although its deposits were even more irregular than those of the Union. When last worked, the Mammoth was producing sulphide ore, the Union both sulphide and oxidized ore, but chiefly the latter; while the Billy has been of value as a source of lean cupriferous pyrite, to mix with the oxide ores of the Union. By judicious combining of the products of the three mines a self-smelting ore was obtained, so that in two years' run the Sasco smelter used no flux. The sulphide ore may run as low as 2.5 per cent. of copper, although many masses showing 10 or 15 per cent. have been found. One 6-ft. breast of nearly solid chalcopyrite was worked out in the Mammoth. The oxide ores carry from 8 to 12 per cent. of copper on the average, while the lean pyrite ores of the Union may have less than 1 per cent. Except some isolated pockets, all the ore is low in silver, averaging not more than an ounce to the ton. It was in the oxidized portion of these separate pockets of lead-silver ore that the first work in the district was done, the assays often showing from 15 to 17 oz. of silver. The notion that these deposits as a whole are valuable for both copper and silver, and that the oxidized ores are characteristically rich in silver, is incorrect.

Shape and Distribution.—The ore-bodies are very irregular, the smaller ones forming pockets and bunches, often roughly lenticular in shape. The larger deposits are more like pipes or chimneys several hundred feet in vertical dimensions and pinching and swelling in diameter from 20 to 100 ft. or more. Horses of more or less garnetized marble are often seen. Tongues, stringers, and other off-shoots of ore into the marble wall-rock are common.

In general, the ore is found on the contact of the marble with either alaskite-porphyry or biotite-granite-porphyry. Parts of an ore-body may be 100 ft. or more from an intrusive rock; but some portion of every ore-mass is close to the contact with

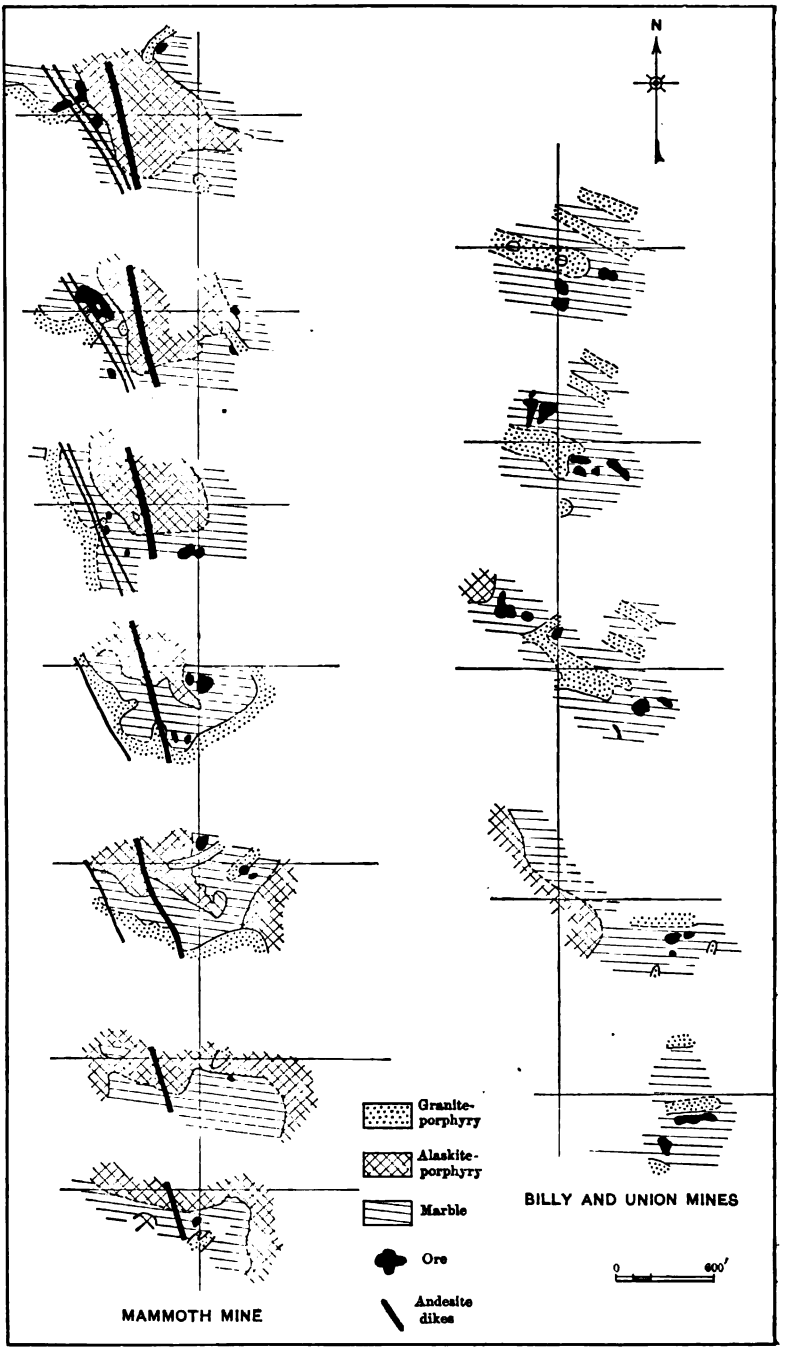


FIG. 18.—GENERALIZED HORIZONTAL SECTIONS ON THE LEVELS OF THE MINES OF THE IMPERIAL COPPER CO. THE VERTICAL AND HORIZONTAL LINES MARK THE INTERSECTION OF VERTICAL PLANES, SO THAT CORRESPONDING POINTS MAY BE EASILY FOUND. LEVELS ARE AT APPROXIMATELY 100-FT. INTERVALS.

one of the two above-named igneous rocks. Usually there is a 5- to 10-ft. band of garnet rock between ore and porphyry. The relation of the metalliferous deposits to the biotite-granite-porphyry and the alaskite-porphyry can best be understood by references to the sketches of the underground geology in Fig. 18. These maps are based on work done by me, except those of the three upper levels of the Mammoth mine, which were made by the engineers of the Imperial Copper Co. The black patches indicating ore do not differentiate between ore worked out and ore in sight, nor do they show all of the ore-bodies or their actual size. They do serve, however, to give an accurate idea of the shape and distribution of these bodies. Several interesting facts brought out by these maps are referred to under the discussion of the origin of the ores.

Contact-metamorphic copper-ores are found in other places in this district, but the only ones that have been extensively developed are those on the property of the Imperial Copper Co. described above. The Young America has a 500-ft. shaft on the contact; and while the price of copper was high this was leased and worked at considerable profit, but in later years the company has confined its operations to prospecting the disseminated ores in porphyry. In the NW. end of the lime belt, copper-stains have been found in the garnet caps, but what exploration has been done has failed to reveal any considerable amount of ore. The El Tiro shaft (Fig. 8) got most of its output from ore in the alaskite. The contact-metamorphic ores, therefore, are most important where biotite-granite is well developed.

Ore- and Gangue-Minerals.—The unaltered ore-bodies consist chiefly of cupriferous pyrite and chalcopyrite, with small and irregular amounts of sphalerite and galena, in a gangue of garnet, quartz, calcite, and wollastonite. The oxidized ores are a somewhat friable mixture of earthy red hematite, copper oxide (probably cuprite), and clayey decomposition-products of the gangue-minerals. Other minerals found in minor amounts are given below.

Bornite, rare and probably secondary.

Chalcocite, rare.

Native copper, found in some oxidized ores, but not important.

Azurite, malachite, and chrysocolla, fine specimens are said to have been found in the early days of the camp, and stains are now seen.

Molybdenite, in small stringers and one large pocket in the sulphide ores.

Siderite, a few small crystals found.

Magnetite, reported from one place in the Mammoth, but both it and specularite practically lacking here.

Barite, platy masses in stringers, but not generally mixed through the ores.

Fluorite, green in color, was found in considerable quantity in an old lead-silver pit near the Mammoth, but in alaskite-porphyry and not in the contact-ores.

Wulfenite, brownish plates in same body as the fluorite.

Cerussite in silky crystals, hershelite, and earthy mixtures of lead and zinc carbonates in some parts of the oxidized ores.

The ore-minerals are definitely later than most of the gangue-minerals, and occur commonly as veinlets in the garnet-quartz-calcite mass, or disseminated through it. The formation of quartz seems to have extended over a considerable period, beginning with the crystallization of garnet and lasting until after the precipitation of the ore-minerals. There has been some post-ore deposition of calcite. These facts coincide with the paragenesis of contact-ores in general.[20] A few polished specimens studied in reflected light showed that while pyrite and chalcopyrite are often intergrown with sphalerite, neither of them is found in the galena observed in the same specimens; moreover, this galena seems to be younger than the sphalerite. The significance of this fact is discussed in connection with the origin of the lead-silver ores.

Secondary Enrichment.—The oxidized zone in the contact-ores is, in general, about 250 ft. deep, but its lower limit is rather irregular, being influenced by the lines of fracture and shearing that are so abundant. The ore in this zone is considerably enriched by the leaching-out of the sulphur and some of the iron- and gangue-minerals. A sulphide ore carrying from 2 to 3

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[20] *Engineering and Mining Journal*, vol. xc., No. 11, pp. 513 to 515 (Sept. 10, 1910).

per cent. of copper will show 10 or 12 per cent. in the oxidized zone. This process of leaching has not affected the copper. There is little migration and secondary precipitation of this metal in the ores, and oxide bodies pass to original sulphides with no transition-zone of secondary sulphides. This fact, taken in connection with the formation of secondary chalcocite in the porphyry-ores of this same district, is a further confirmation of the idea advanced by D. C. Bard that calcitic gangue has a tendency to prevent the downward migration of copper.[21] The results of some experiments along this line by T. B. Welch and myself are now in preparation, and show that calcite does have a precipitating effect on copper sulphate. A puzzling feature of the Silverbell ores, however, is the small amount of carbonate in the oxidized zone. Possibly the larger amounts of manganese present may account for the excess of copper oxides, but experiments to test this hypothesis have thus far given no satisfactory results.

2. *Disseminated Copper-Ores.*

Pyrite, more or less copper-bearing, is common throughout the fresh alaskite-porphyry, the alaskite, and the granite, and its weathering-products have stained these rocks red or buff at the surface. Occasionally, green copper-stains are seen, but usually the copper has gone into solution and been precipitated below. Extensive prospecting has been done, first by test-shafts and later by churn-drilling, in an effort to discover a workable deposit of disseminated copper-ore in these rocks. The results of these operations were not available as far as definite figures go, but some facts of importance in regard to the geology were obtained.

These disseminated ores can be of value only where secondary enrichment has taken place. Chalcocite of secondary character can be found anywhere in the camp, but not always sufficiently rich or extensive to make a workable deposit. Prospecting in the alaskite-porphyry, although showing some rich streaks, was unsatisfactory; and the Imperial Co. has turned its attention to the coarse alaskite, with much better results. One of the best holes showed 60 ft. of 2-per cent. ore. A microscopic examination of some specimens shows that the

[21] *Economic Geology*, vol. v., No. 1, p. 59 (Jan., 1910).

ore occurs as minute stringers and specks of chalcocite in the alaskite. It usually has a core of unaltered pyrite, and the pannings from the drill-samples show pyrite cubes, coated with chalcocite, and some pseudomorphs of chalcocite after pyrite.

The depth to which the oxidized zone extends is extremely variable, but generally not more than 100 ft., and the zone of secondary sulphides is from 50 to 60 ft. thick. Some holes have gone through a chalcocite zone, into an oxidized zone, and then back to chalcocite; and it is very common to pass through bright fresh pyrite before reaching the chalcocite. Ore shipped from the El Tiro shaft was from a zone of much broken alaskite and consisted of chalcocite, coating stringers of pyrite. Here oxidation had extended to the unusual depth of more than 300 ft., and secondary sulphides were found down to the 500-ft. level. A cave-in stopped mining-operations before much was known of the geology. The El Tiro ore was concentrated, and the tables are said to have shown a mixture of chalcocite with cuprite and native copper.

The alaskite probably shows richer concentrations of chalcocite than the porphyry, not because of higher original copper-content, but because its coarseness of grain and numerous joints have been more favorable to the processes of secondary enrichment. The varying depths of the chalcocite zones, the alternation with oxidized material, and the presence of pyrite above some of the chalcocite, are believed to indicate that this secondary concentration has been made extremely irregular by the influence of sheared and fissured areas. This seems to be confirmed by the occurrence at the El Tiro. If it be true, mining these disseminated ores will be an unusually difficult matter, and very careful exploration will be necessary to allow for the sudden changes which will be found in the boundaries of the chalcocite zone.

All the drilling in the Young America property has been confined to the alaskite-porphyry, but whether with more favorable results than the Imperial Co. obtained in the same rock, is unknown.

3. *Origin of the Copper-Ores.*

That contact-metamorphic ores are the result of magmatic emanations from the accompanying intrusive is too generally

accepted to need further discussion here. The question of importance at Silverbell is, which igneous rock caused the ore-deposition?

In the early days of the camp no distinction was made between alaskite-porphyry and granite-porphyry, and prospecting was done indiscriminately along the contact of the marble with either of these two rocks. When the types were distinguished, the alaskite-porphyry was considered as the source of the mineralization; but B. L. Smith, the present superintendent, was struck by the fact that granite-porphyry was often the igneous rock nearest the ore. A reference to the maps in Fig. 18 will show that except for some lean pyrite bodies on the west that are in the Billy mine, all the ores of the Union are associated with granite-porphyry, occurring most often near the end of a large tongue of this rock. In the Mammoth mine alaskite-porphyry is the chief rock on the north of the marble, and granite-porphyry on the south and west, and ore-bodies are found on both contacts. The north, however, is not exclusively alaskite-porphyry, but shows tongues of granite-porphyry, and usually any ore-body, if followed vertically from one level to another, will somewhere come in close contact with a mass of granite-porphyry. When we consider, in addition, the greater amount of contact-metamorphism and ore-deposition apparent in that part of the district where biotite-granite is best developed, we cannot escape the conclusion that it was the most important source of mineralizing solutions. That it was not the only source is shown by the lean pyrite bodies in the Billy, where alaskite-porphyry is the only intrusive.

The magmatic solutions that caused the contact-metamorphism and the ore-deposits probably were not given off exclusively by the igneous rocks now seen. It is believed that these emanations came from the depths of the magma and rose along the intrusive. This seems most in accord with the alteration undergone by the alaskite- and granite-porphyries, and with the fact that the pyrite of the igneous rocks is a later impregnation and not pyrogenetic. The origin of the copper-ores, therefore, was most probably as follows: Beginning with, and continuing after, the intrusion of the alaskite-porphyry, hot waters rose along this rock from the magma below. These waters were first rich in CO_2, then siliceous and ore-bearing;

they first sericitized the igneous rocks, then silicified them, at the same time metamorphosing the limestone to garnet rock; and, lastly, they impregnated the porphyry with cupriferous pyrite, and deposited pyrite and chalcopyrite in the contact-zone, both by filling minute pores and fissures and by replacement. Before the alaskite-porphyry was completely solidified, and while these emanations were still being given off, biotite-granite was intruded from the parent magma, thus opening up channels along which the emission of the above-mentioned hot waters took place with renewed activity. It is, therefore, logical to expect to find contact-ores near the alaskite-porphyry, but a more favorable place, and one where the ores may be considerably richer, is where tongues of granite-porphyry are present.

The above explains the origin of the cupriferous pyrite in the alaskite-porphyry and granite as a deposition from solutions coming up from below, but it does not account for the pyrite in the coarse alaskite which has furnished the favorable chalcocite-prospects. For this two views are possible: (1) the solutions from the porphyry may have penetrated the alaskite and thus impregnated it with pyrite; or (2) ore-bearing solutions may have come up along the coarse alaskite shortly after its intrusion, and entirely independent of the porphyry. If the first view is correct, the chalcocite-deposits may be expected to extend not far from the contact with the porphyry. Thus far, all prospecting has been done near the contact, and the possibility of this limitation in extent may well be kept in mind.

In view of the fact that the coarse alaskite does not show much of the silicification characteristic of the action of the waters rising along the porphyry and granite, it may well be that these emanations did not affect this rock. In that event the coarse alaskite would owe its mineralization to hot solutions arising from below shortly after its intrusion, and entirely independent of the porphyry, and its chalcocite-deposits would not be limited to the contact. As this rock is doubtless derived from the same general magma, there is no apparent reason why its intrusion should not have been followed by ore-bearing solutions similar to those associated with the porphyry.

4. *Lead-Silver Ores.*

As has been noted above, several pockets of lead-silver ore were worked in the vicinity of the Mammoth mine in the early history of the district. At the present time the small amount of silver found in the copper-mines of the Imperial Co. is in distinctly separate stringers and bunches.

East of the property of this company residents of Silverbell have taken up a number of claims on a fissure-vein of lead-silver ore in the quartz-porphyry (Fig. 8). In this vicinity are dikes of rhyolite-porphyry, and a more basic rock called mica-andesite by Prof. C. F. Tolman, Jr. The vein is the filling of a fault-fissure that strikes nearly N–S., dips 80° W., and has been followed more than 500 ft. along the strike, with one spur running a short distance to the east. This is undoubtedly a distinct fault-fissure; for it cuts off a rhyolite dike, and is filled with brecciated material. Some post-mineral movement has taken place along the western wall. A shaft about 100 ft. deep, sunk on the vein, shows that the deposit is really a cementing of a brecciated zone which is about 5 ft. wide at this point. The ore is galena in a gangue of fluorite, calcite, quartz, and fragments of quartz-porphyry. Considerable cerussite and anglesite occur in the upper part, and native silver and cerargyrite are said to have been found by the prospectors. There are a few green stains of copper carbonate, but the outcrop of the vein is for the most part not marked by discoloration. The owners of the claims report very favorable assays, and are planning active exploration.

A thin section of one of the wall-rock fragments found in the vein shows a quartz-porphyry now much altered. Only a few patches of the original ground-mass remain, the rest being a mass of fine-grained quartz and some larger fluorite crystals. The mica and the feldspars have largely disappeared also, but the quartz phenocrysts and unaltered ground-mass bear a close resemblance to the fresh quartz-porphyry. Having in mind the fact that the copper-ores of Silverbell were first worked for silver, several engineers have expressed the opinion that this ore-body also would probably turn to copper in depth. This is reasoning on a false analogy, for this lead-silver vein is in the quartz-porphyry, and therefore is later

than the contact copper-ores, and due to an entirely distinct period of mineralization.

In this connection it is interesting to consider the origin of the occasional silver-ores found in the contact copper-deposits. None of these was being worked when I visited the district, but Mr. Smith was emphatic in saying that high silver-values are sharply localized. It is possible that the same mineralization which caused the vein just described also formed the stringers of lead-silver in the contact-deposits. I could not get sufficient data on the shape and mineral content of the ores in question to justify a definite conclusion, but the above suggestion is supported by the finding, in an old lead-silver working near the Mammoth shaft, of great quantities of fluorite, which is unknown in the contact-ores proper, and by the evidence of polished specimens, that, while sphalerite is clearly associated with the copper-minerals, galena is not.

In any event, there can be no doubt that there is no apparent reason to expect the lead-silver prospects on the east to change to copper-ore in depth.

IX. Bibliography.

1. *References to the Geology of Arizona.*

Barney, W. G., The Silverbell Mountains, Arizona, *Engineering and Mining Journal*, vol. lxxviii., No. 19, pp. 755 to 756 (Nov. 10, 1904).

Blake, W. P., The Geology of Arizona, Report of the Governor of Arizona for year ending June 30, 1904, pp. 66 to 72.

Comstock, T. B., Geology and Vein-Phenomena of Arizona, *Trans.*, xxx., 1038 to 1101 (1900).

Lindgren, Waldemar, The Copper Deposits of the Clifton-Morenci District, Arizona, *Professional Paper No.* 43, *U. S. Geological Survey* (1905).

Lindgren, W., L. C. Graton and C. H. Gordon, Ore Deposits of New Mexico, *Professional Paper No.* 68, *U. S. Geological Survey* (1910).

Ransome, F. L., Geology of the Globe Copper District, Arizona, *Professional Paper No.* 12, *U. S. Geological Survey* (1903).

——— Geology and Ore Deposits of the Bisbee Quadrangle, Arizona, *Professional Paper No.* 21, *U. S. Geological Survey* (1904).

Tolman, C. F., Jr., Disseminated Chalcocite Deposits at Ray, Arizona, *Mining and Scientific Press*, vol. xcix., No. 19, pp. 622 to 624 (Nov. 6, 1909).

——— Copper Deposits of Silverbell, Arizona, *Mining and Scientific Press*, vol. xcix., No. 22, pp. 710 to 712 (Nov. 27, 1909).

——— The Miami-Inspiration Ore-Zone, *Mining and Scientific Press*, vol. xcix., No. 20, pp. 646 to 648 (Nov. 13, 1909).

——— Southern Arizona Copper Fields, *Mining and Scientific Press*, vol. xcix., No. 11, pp. 356 to 360 (Sept. 11, 1909); No. 12, pp. 390 to 393 (Sept. 18, 1909).

Wheeler, G. M., *Geographical and Geological Surveys West of the One Hundredth Meridian.* Final Report, vol. iii. (1875).

Weed, W. H., The Ray Copper Mining District, Arizona, *Mining World*, vol. xxxiv., No. 2, pp. 53 to 56 (Jan. 14, 1911).

2. *Special Papers on Contact-Metamorphism.*

Barrell, Joseph, Physical Effects of Contact Metamorphism, *American Journal of Science*, ser. 4, vol. 13, No. 76, p. 279 (Apr., 1902).

Beck, Weed, *The Nature of Ore Deposits*, pp. 582 to 606 (1905).

Beyschlag, F., P. Krusch and J. H. L. Vogt, *Der Lagerstätten der Nutzbaren Mineralen und Gesteine*, pp. 344 to 370 (Stuttgart, 1910).

Blake, W. P., Copper-Ore and Garnet in Association, *Trans.*, xxxiv., 886 to 890 (1903).

Kemp, J. F., Ore Deposits at the Contacts of Intrusive Rocks and Limestones, and Their Significance as Regards the General Formation of Veins, *Economic Geology*, vol. ii., No. 1, pp. 1 to 13 (Jan.-Feb., 1907).

Keyes, C. R., Garnet Contact Deposits of Copper and the Depths at which They Are Formed, *Economic Geology*, vol. iv., No. 4, pp. 365 to 372 (June-July, 1909).

Klockmann, F., Uber Kontaktmetamorphe Magnetitlagerstätten, ihre Bildung und systematische Stellung, *Zeitschrift für praktische Geologie*, vol. xii., pp. 73 to 85 (Mar., 1904).

Lindgren, W., Character and Genesis of Certain Contact-Deposits, *Trans.*, xxxi., 226 to 244 (1901).

Stutzer, O., Kontaktmetamorphe Erzlagerstätten. Zusammenstellungen und Betrachtungen, *Zeitschrift für praktische Geologie*, vol. xvii., pp. 145 to 155 (Apr., 1909).

Van Hise, C. R., A Treatise on Metamorphism, *Monograph XLVII., U. S. Geological Survey* (1904).

Vogt, J. H. L., Problems in the Geology of Ore-Deposits, *Trans.*, xxxi., 125 to 169 (1901).

Weed, W. H., Ore-Deposits near Igneous Contacts, *Trans.*, xxxiii., 715 to 746 (1902).

3. *Descriptions of Contact-Metamorphic Districts of General Importance.*

Barrell, Joseph, Geology of the Marysville Mining District, Montana, *Professional Paper No.* 57, *U. S. Geological Survey* (1907).

Bayley, W. S., The Eruptive and Sedimentary Rocks on Pigeon Pt., Minnesota, and their Contact Phenomena, *Bulletin No.* 109, *U. S. Geological Survey* (1893).

Beals, Wm., Jr., The Seven Devils Mining District, Idaho, *Engineering and Mining Journal*, vol. lxix., No. 12, pp. 345 to 346 (Mar. 24, 1900).

Bergeat, A., Der Granodiorit von Concepcion del Oro im Staate Zacatecas (Mexiko) und seine Kontaktbildungen. *Neues Jahrbuch für Mineralogie, Geologie, und Paläontologie*, vol. xxviii., pp. 421 to 569 (1909).

Boutwell, J. M., Arthur Keith, and S. F. Emmons, Economic Geology of the Bingham Mining District, Utah, *Professional Paper No.* 38, *U. S. Geological Survey* (1905).

Crosby, W. O., Limestone-Granite Contact-Deposits of Washington Camp, Arizona, *Trans.*, xxxvi., 626 to 646 (1905).

Harder, E. C., The Taylor Peak and Whitepine Iron-Ore Deposits, Colorado, *Bulletin No.* 380, *U. S. Geological Survey*, pp. 188 to 198 (1909).

Harker, A., and J. E. Marr, The Shap Granite, and the Associated Igneous and Metamorphic Rocks, *Quarterly Journal of the Geological Society of London*, vol. xlvii., pt. 3, pp. 266 to 326 (Aug. 1, 1891).

Kemp, J. F., Copper-Deposits at San Jose, Tamaulipas, Mexico, *Trans.*, xxxvi., 178 to 203 (1905).

——— Review of *Bulletin* by Leith and Harder on Iron Springs District, *Economic Geology*, vol. iv., No. 8, pp. 782 to 791 (Dec., 1909).

Kemp, J. F., and C. G. Gunther, The White Knob Copper-Deposits, Mackay, Idaho, *Trans.*, xxxviii., 269 to 296 (1907).

Kellogg, L. O., Sketch of the Geology and Ore Deposits of the Cochise Mining District, Cochise County, Arizona, *Economic Geology*, vol. i., No. 7, pp. 651 to 659 (July–Aug., 1906).

Knopf, Adolph, Geology of the Seward Peninsula Tin Deposits, Alaska, *Bulletin No.* 358, *U. S. Geological Survey* (1908).

Leith, C. K., and E. C. Harder, Iron Ores of the Iron Springs District in Southern Utah, *Bulletin No.* 338, *U. S. Geological Survey* (1908).

Lindeman, E., Preliminary Report on the Iron Ore Deposits of Vancouver and Texada Islands, *Canadian Department of Mines, Mines Branch, Summary Report*, 1907–08, pp. 35 to 43 (1908).

Paige, Sidney, The Hanover Iron-Ore Deposits, New Mexico, *Bulletin No.* 380, *U. S. Geological Survey*, pp. 199 to 214 (1909).

Prescott, Basil, Occurrence and Genesis of the Magnetite Ores of Shasta Co., California, *Economic Geology*, vol. iii., No. 5, pp. 465 to 480 (Sept., 1908).

Ransome, F. L., The Yerington Copper District, Nevada, *Bulletin No.* 380, *U. S. Geological Survey*, pp. 102 to 119 (1909).

Smith, George, Garnet-Formations of the Chillagoe Copper-Field, North Queensland, Australia, *Trans.*, xxxiv., 467 to 478 (1903).

Spurr, J. E., and G. H. Garrey, Ore-Deposits of the Velardeña District, Mexico, *Economic Geology*, vol. iii., No. 8, pp. 688 to 725 (Dec., 1908).

Stutzer, O., Die kontaktmetamorphen Kupfererzlagerstätten von White Horse in Yukon (Canada), *Zeitschrift für praktische Geologie*, vol. xvii., pp. 116 to 121 (1909).

Turner, H. W., Notes on Contact-Metamorphic Deposits in the Sierra Nevada Mountains, *Trans.*, xxxiv., 666 to 670 (1903).

Weed, W. H., The Elkhorn, Montana, Mining District, *Twenty-second Annual Report, U. S. Geological Survey*, pt. ii., pp. 419 to 510 (1900–01).

Yung, M. B., and R. S. McCaffery, Ore-Deposits of the San Pedro District, New Mexico, *Trans.*, xxxiii., 350 to 362 (1902).

4. *Miscellaneous.*

Blake, W. P., Superficial Blackening and Discoloration of Rocks Especially in Desert Regions, *Trans.*, xxxv., 371 to 375 (1904).

Clarke, F. W., The Data of Geochemistry, *Bulletin No.* 330, *U. S. Geological Survey* (1908).

Iddings, J. P., *Igneous Rocks* (New York, 1909).

Harker, Alfred, *The Natural History of Igneous Rocks* (New York, 1909).

Lawson, A. C., *Bulletin of the Department of Geology, University of California*, vol. iii., No. 17, p. 388 (1902–04). (Refers to Quartz Dikes.)

Spurr, J. E., Ore Deposits of the Silver Peak Quadrangle, Nevada, *Professional Paper No.* 55, *U. S. Geological Survey* (1906).

——— Igneous Rocks as Related to the Occurrence of Ores, *Trans.*, xxxiii., 238 to 340 (1902).

Stevens, H. J., *The Copper Handbook*, vol. vii. (1907).

The Bearing of the Theories of the Origin of Magnetic Iron-Ores on Their Possible Extent.

BY FRANK L. NASON, WEST HAVEN, CONN.

(New York Meeting, February, 1912.)

In the year 1904 an eminent Swedish geologist prepared a report on the iron-ore reserves of the world. His estimates follow:

Countries.	Tons.
United States,	1,100,000,000
Great Britain,	1,000,000,000
Germany,	2,200,000,000
Spain,	500,000,000
Russia and Finland,	1,500,000,000
France,	1,500,000,000
Sweden,	1,000,000,000
Austria-Hungary,	1,000,000,000
Other countries,	1,200,000,000
Total,	10,000,000,000

The iron-ore allotted to the United States by the Swedish geologist is barely sufficient to last 20 years at the present rate of consumption.

On the other hand (according to Edwin C. Eckel,[1] from whom the above and following statistics are quoted), there is hardly need for worry on the part of our iron industries for the next two centuries or more.

Upon the basis of careful work by the U. S. Geological Survey, in which Mr. Eckel has taken an important part, the following estimates are definitely, but by no means finally, given for the United States:

District.		Tons.
Lake Superior District,		1,500,000,000 to 2,000,000,000
Alabama,	red hematite,	1,000,000,000
	brown hematite,	725,000,000
Georgia,	red hematite,	200,000,000
	brown hematite,	125,000,000
Tennessee,	red hematite,	600,000,000
	brown hematite,	225,000,000
Virginia,	red hematite,	50,000,000
	brown hematite,	300,000,000
Total,		4,725,000,000 to 5,225,000,000

[1] *Mineral Resources of the United States*, pp. 67 to 102 (1906).

These figures deal almost exclusively with red and brown hematites, which, as Mr. Eckel justly remarks, can be estimated with a possible error of from 5 to 10 per cent. Except in the Michigan and Wisconsin fields, magnetites do not enter at all into Mr. Eckel's estimate. Moreover, it is confined to known mines and districts, usually well exploited, and does not cover the possibilities of new fields in the States named, or of similar fields in other States. Mr. Eckel says:

"If to this we add the ores occurring in the deeper levels in the States named, and also the red and brown ores of Maryland, West Virginia, and Kentucky, and the magnetic ores of the other Southern States, it is probably fair to assume that the total Southern ore reserve will amount to very nearly 10,000,000,000 tons, or five times that credited to the Lake Superior district."

On page 78 Mr. Eckel remarks:

"The amount of Clinton ore in any area can probably be determined by a geologist within 5 or 10 per cent., and estimates of red ore reserves can, therefore, be made by competent men with a degree of accuracy impossible in dealing with magnetites, hematites, or brown ores of other districts."

To make a final quotation from Mr. Eckel's sane and comforting paper:

"It is probable that careful exploratory work will develop magnetic iron-ores in New York, New Jersey, and Pennsylvania in quantities far in excess of anything usually considered possible in these States."

In my opinion Mr. Eckel could have included in his list not only most of the New England States, but the Appalachian States as well.

From the above citations, we may rest assured that an iron-ore famine does not immediately threaten the United States. The accuracy with which tonnage-reserves of the Clinton iron-ores can be estimated will hardly be disputed by any geologist. To show that deposits of magnetic iron-ores, whatever their origin, can be almost as accurately estimated, and that, with these ores added, Mr. Eckel's figures of possibly ten to twenty billion tons can probably be doubled, is the main object of this paper.

The theories as to the origin of magnetic iron-ores may be grouped as follows:

I. Primary or Cotemporaneous (*i. e.*, sedimentary origin).

II. Secondary (practically vein-formation).

III. Igneous (*i. e.*, segregation from molten magma).

Evidently, this classification is not perfect, in the sense that the three theories are mutually exclusive; but it will serve my purpose.

Given, or assuming, that the origin of magnetic iron-ores be ascribed to either of the above divisions, what bearing will the assumption have: (1) on the areal extent of any given occurrence; (2) on the ability of a competent geologist to estimate the probable volume of magnetic iron-ores?

In short, whether the magnetic iron-ores originated as sediments, as secondary, or as igneous deposits, are there any fixed relations between ore and country, or any characteristics of the ore-bodies, *per se*, which will enable a geologist to limit the extent of a given magnetic iron-ore field with the close accuracy possible with the Clinton ores?

This is the burden of the analysis which follows.

I. Primary or Cotemporaneous Origin of Magnetic Iron-Ores.

This theory means that the magnetic iron-ores are a sedimentary rock-formation occurring in an uninterrupted chronological sequence. If it be accepted, what bearing does it have on the extent of the magnetic iron-ores? Does its careful study afford a means of predicting, within economic limits, the extent of these ores?

The sedimentary theory affirms that the rocks inclosing the ores were first a series of sediments: limestones, pure to highly impure; interstratified sands, fine to coarse-grained, calcareous or argillaceous; clays, calcareous and arenaceous, etc.

In this series may be interpolated one or more beds of nearly-pure iron-ore, either chemical or mechanical in origin, and one or more beds of impure to very impure iron-ore, of like origin. A railroad-cutting through earth layers, a section of a clay-bank and its overburden, or a rock-cutting or natural rock-section may show the sequence of deposition, and also how any given stratum thins and thickens, or changes along its strike from pure to impure, or even from slate to limestone or from limestone to sand or slate. Such a change from pure to impure ore, and *vice versa*, can be easily imagined, and is actually observed in Red Mountain, Birmingham, Ala.

Upon the deposition of a ferruginous series of rocks, may, and probably will, follow a series of rocks so free from iron as to be classed as non-ferruginous. We thus have foot- and hanging-wall rocks developed, as known to the miner. Between

these walls will be the "rich" or "lean" ore, and rock which contains so little iron as not to be classed as ore at all.

Recurring to the actual formative period, it is important to note that the variation in the iron-content of the iron-ore series must be referred to the shore-line of the sea. The iron-series may be forming for miles along the shore. This will give one dimension of the iron-ore series, the length on strike; the second dimension, width, *i. e.*, distance sea-ward, will be determined by the period of time during which either subsidence has taken place, or the sea has caten inland from the first shore-line.

To anticipate, somewhat, our conclusions: judging both by the iron-bearing gneisses and the magnetites themselves, the shore-line of the Archæan sea reached, on our Atlantic seaboard, from Ungava bay to the State of Alabama, a distance of over 2,400 miles. In New Jersey, these gneisses outcrop across their strike, a distance of 18 miles; and workable deposits of magnetic iron-ore occur not only on the extreme margins, but at many points between. On the sedimentary theory, the exposed rocks of the iron-bearing period exist inland at least 18 miles, with no diminution of iron except in variable amounts. Farther in New Jersey, to cite a single example out of many possible ones, there is a succession of superimposed strata, covering over 1,000 ft. of thickness at right angles to bedding-planes. This instance can be supplemented and fortified by well-known occurrences of magnetic iron-ores, and their peculiar accompanying "gray gneisses," in SE. New York, in the Adirondack region, and in southern Canada from Port Arthur to the St. Lawrence river, in eastern Ontario.

On the assumption of sedimentary origin, we have, then, ocular proof in New Jersey that the geological period of the formation of magnetic iron-ores covered from West Point, on the Hudson river, to the Delaware, a shore-line 84 miles long, and that this shore-line advanced inward 18 miles.

Further, making no assumptions based on the entire thickness of the magnetite-bearing gneisses, we know, from actual superimposed beds of magnetic iron-ore, that one ore-series is more than 1,000 ft. thick; that in this series are five workable deposits of ore; and that these deposits average more than 10 ft. in thickness from foot- to hanging-wall.

From the "primary," "cotemporaneous," or "sedimentary" origin of magnetic iron-ores, as here stated, it follows as a corollary that the iron-ores are conformable, that is, more or less parallel to their inclosing rocks, and that probably the iron-ore stratum will be more or less co-extensive with the inclosing rocks, after the manner of coal-seams or beds.

Variation in Sedimentation.

Sedimentation, or cotemporaneous deposition of iron-bearing strata, may be effected by mechanical or chemical processes, or both together. One part of a given stratum may be wholly clastic in origin; another part may be wholly chemical; or any given part, or the entire stratum, may combine the two factors in varying proportions. Which of the two has been the dominant one, has an important bearing on the possibility of estimating the extent of the magnetic iron-ores.

1. *Mechanical, or Clastic, Origin.*—This requires three conditions: (1) an open sea-front; (2) granitic, gneissic, or eruptive rocky shores, either disintegrated or solid or both; and (3) iron-ore in the rocky shore-line—and this iron-ore must be in solid grains or masses, as magnetite, hematite, or limonite.

An open sea is an indispensable assumption: otherwise there would be no waves to "pan" the lighter sand, etc., leaving the heavier iron-ore in windrows on the beach. A shore of granites, gneisses, or eruptives must be assumed, since limestones, clay-slates, sandstones, etc., rarely carry grains of iron-ore in appreciable quantities.

The iron must exist in these rocks in the form of magnetite, hematite, or limonite, in grains or masses, otherwise there could be no mechanical concentration or panning. The extent of the deposit of clastic iron-ore would depend directly on the percentage of iron in grains and masses in the shore-line.

It is a well-known fact that grains of magnetite and hematite are not easily dissolved by natural agencies. The occurrence of magnetite in sea- and lake-sands; in mountain streams and on inland plains; the occurrence of grains and crystals of magnetite, still highly magnetic, in the rotted gneisses and granites of the non-glaciated regions, etc.—give sufficient evidence in this line, and lend great plausibility to the theory of the clastic origin of magnetites.

2. *Chemico-Sedimentary Origin.*—For this process, we must assume conditions the very reverse of those assumed for the clastic phase. Instead of an open sea, it requires closed lagoons, low, generally or periodically flooded marshes or great inland lakes or seas; since chalybeate waters, however heavily burdened, pouring into an open sea would be so widely dissipated that no economic ore-body would be formed.

Iron is one of the most widely and abundantly distributed of the elements. According to Dr. Clarke, metallic iron constitutes 4.49 per cent. of the earth's crust. Therefore, under normal conditions in waveless, inclosed or sheltered waters, soluble iron would be leached to the extent of 4.49 per cent. of the rock-mass, of whatever nature, and precipitated more or less evenly and regularly over the entire bottom of the lagoon. While the "average per cent." of Dr. Clarke includes the entire rock-crust of the earth, it is evident that a large part of that crust is iron-free. This rather militates in favor of our assumption. Since the basis of our discussion is an actual ore-deposit, if the deposit be ascribed to chemical solution and precipitation, the decomposing rocks feeding the deposit must have had at least the 4.49 per cent. average of Dr. Clarke.

The granites and gneisses would contribute only their soluble iron in biotites, hornblendes, etc. The shores being tideless and waveless, magnetite, hematite, garnet, and silica would remain *in situ;* iron, lime, magnesia, and alkalies would be leached out and carried into lagoons, there to be precipitated or in part to remain in solution. It is, however, possible to imagine that the readily-soluble minerals of gneiss, granite, or eruptives, and, to a greater or less extent, alumina, might be wholly eliminated. There would thus be left the insoluble magnetite and hematite, in grains or even in large masses, to form "residual" beds; and these would be supplementary to the deposits strictly chemical in origin. It is hardly necessary to state in detail how chemical and mechanical force could each play a part in the same stratum.

Economic Bearing of the Sedimentary Theory.

The Clinton iron-ores are regarded as cotemporaneous sediments, and likewise as chemical precipitates. Experience has taught that these ores are nearly coextensive with the contain-

ing rocks. For these reasons, in a given locality, a competent observer can estimate the tonnage within an error of from 5 to 10 per cent. If, therefore, the chemico-sedimentary origin of magnetic iron-ores can be proved, or even shown to be reasonable, there is no reason why a careful study of magnetic iron-ore fields should not permit an equally accurate estimate.

Without going into detail, we must confess that, if the magnetites are of wholly clastic origin, the above margin of from 5 to 10 per cent. is too small to be safe. For a combination of the two origins, chemical and clastic, the chances for accuracy in a given district are about equal with those attending the Clinton ores. In both cases, the fundamental principle is simple.

The magnetic iron-ores are a rock-stratum; this stratum outcrops in rocks of its own age, and these rocks cover thousands of square miles; there is thus no known or imaginable reason why one should limit the extent of a magnetic iron-ore to a few feet in length and a few feet in depth.

Statistics.—Whether or not the true reason for the common skepticism as to our magnetic iron-ore reserves is indicated by the following figures, they are suggestive:

As late as 1880, the Appalachian region produced 5,667,348 tons of iron-ore, or 67 per cent. of the total of 8,487,250 tons. Of this total, 30.02 per cent. was magnetite. In 1910, the total iron-ore production of the United States was about 53,000,000 tons, of which about 5 per cent. was magnetite.

In 1880, magnetic iron-ores were chiefly mined in New York and New Jersey. The ore-shoots then economically workable were small, because all ores carrying less than 50 per cent. of iron were unmarketable. The total product of each mine was small, and the cost high; consequently, mining to great depths was economically impracticable; and a "pinch" usually caused a shut-down. Exploration was limited to rich shoots and moderate depths.

Four years later the great high-grade deposits of Minnesota began their shipments.

These facts may explain why outcrops of magnetic iron-ore are supposed, generally, to cover no more than 6,000 sq. ft., *i. e.*, 600 by 10 ft., and to become exhausted at less than 1,000 ft. in depth on an average. Drop the required iron-content of an

ore from 55 to 35 per cent.; and, to my personal knowledge, there are magnetic iron-ore outcrops from 1 to 7 miles long. In actual thickness of the bed, these bodies vary from 2 to over 100 ft. In extension on the dip, they have been proved less than 1,000 ft. on an average, but in one instance over 3,000 ft., and the boundary not there demonstrated. By the sedimentary theory, then, the magnetic ores, as well as the Clinton, should be given an inclined depth proportionate to the length of outcrops.

II. Secondary Origin of Magnetic Iron-Ores.

On this hypothesis (whatever the nature of the inclosing rocks, igneous eruptives, metamorphosed sediments, or metamorphosed sediments into which sheets of molten eruptives have been thrust or over whose surfaces molten eruptives have been poured, or both), the magnetic iron-ores were deposited after the rocks were formed, and can only have been so deposited by chemical action, that is, the solution and deposition of the iron, by and from circulating waters.

Such waters may have been derived from either or both of two sources. The first is known as "magmatic" water in eruptives, or as "rock-water" and "water of crystallization" in the case of sedimentary rocks. The second is "meteoric" water, entering from the earth's surface into the underground circulation.

As to the source of the iron to be carried by these waters, it has already been pointed out that metallic iron constitutes nearly 4.5 per cent. of the rocks of the earth's crust. A mass of rock 1 mile square and 500 ft. thick weighs about 1,072,200,000 tons. At 4.5 per cent. of metallic iron in such a block, the content of metallic iron will be 48,249,000 tons, equal to about 83,700,000 tons of magnetite. This would furnish material for a pure, solid magnetite ore-bed 1 mile square, and more than 20 ft. thick.

As we are dealing with the same rock-formation as before, but assuming eruptives or a combination of eruptives and sediments, instead of sedimentary origin, the areas in both cases will be coextensive.

Water and iron having been accounted for, the next step is to find room for deposition.

Two cases will arise here, as has already been suggested. The rocks are either sedimentaries, penetrated or overflowed by eruptives, or they are wholly igneous or eruptive.

In the first case, we may suppose that the original sediments were alternating beds of sandstones, shales, and limestones. Limestones are not only readily soluble, but as the limestone is dissolved in iron-bearing waters and carried away, iron is deposited in its place by a simple chemical reaction. The purity of the deposited iron-ore would depend on the purity of the limestone. A pure limestone would be wholly dissolved and replaced by pure iron-ore, while from an impure limestone all silica, alumina, etc., would be left behind, to mingle with the ore.

We are familiar with the fact that a limestone bed is rarely uniform in composition. Lenses or bunches of impure limestone occur with more or less regularity. Hence, if 1 square mile of rock were originally limestone, a vertical section of it, after the lime had been dissolved out and replaced with iron-ore, would show lenses of pure ore passing into strings of pure ore interbanded with lean—just the conditions, in fact, which the magnetic iron-ore miner finds in his mines to-day.

In the second case, where the entire rock-mass is a succession of eruptives, the rock, instead of easily-soluble limestone, would consist of less easily decomposed feldspars, micas, hornblendes, pyroxenes, etc. The end, however, would be practically the same as with the limestones.

The decomposition by water of solid crystalline rocks is abundantly illustrated by the gneissic and granitic rocks of the unglaciated Southern States, where sections of rotted rocks many feet thick are exposed. As to the power of water to dissolve the iron from these rocks, and the later precipitation of this iron, the 725,000,000 tons of minable "brown ore" which Mr. Eckel estimates in Alabama, Georgia, Tennessee, and Virginia alone, is conclusive proof.

Reviewing, briefly, the salient points in the secondary origin of magnetic iron-ores, we find that, whether the rocks containing the ores originated as sediments or eruptives, they cover many thousands of square miles. As is demonstrated in the case of the Clinton ores, the extent of the ores is not limited to the outcrops, but conditions similar to the outcrops may reach

at right angles to the outcrop for thousands of feet, limited only by the extent of circulating water. We find an abundant water-supply in magmatic, rock, or metoric water. We find channels through which this water can readily circulate. We find an abundance of rocks and minerals which are dissolved or decomposed by water. We find in the rocks themselves an abundant source of iron and in a readily-soluble form.

Therefore we can but conclude that, as in the case of sedimentary origin, so in the case of eruptives there is no visible reason why the magnetic iron-ores should not be coextensive with the special rock-formation, as is the case with the Clinton iron-ores.

III. Igneous Origin of Magnetic Iron-Ores.

This theory appeals to the well-known fact that crystals and crystalline grains of magnetite and hematite are constituents of nearly every crystalline eruptive. The percentage of metallic iron will exceed 4.5 per cent. We have already seen that this means an equivalent of 83,700,000 tons of magnetite in a square mile of rock 500 ft. thick. The problem is, to account for the assembling, not of all, but of a small part of this magnetite in the form of workable ore-bodies.

The principle involved is very simple and can be readily observed by every smelter of iron- or copper-ores. Fragments of matter held in mechanical suspension in a fluid or semi-fluid of lower specific gravity will settle and gather, so long as the suspending medium remains sufficiently fluid.

In copper-smelting, slag and matte, intimately mixed, are drawn from the furnace into pots. So long as the slag remains fluid the matte settles and collects in the bottom of the pot. When the slag becomes viscous in cooling, suspended grains of matte will be fixed in position, and unable to enter the main mass of matte. To apply this principle to the segregation of magnetic iron-ores by the igneous theory we have only to assume that all igneous rocks were at one time more or less fluid, and that they retained their fluidity for a long period. Moreover, we know that magnetite and hematite occur in all eruptives, and that both are nearly twice as heavy as the rock-mass in which they occur.

Therefore, according to this theory, the igneous rocks con-

taining the magnetic iron-ores are but a gigantic slag-pot, from which the matte, in this case iron-ore, has, or may have, settled. One might jump to the conclusion, then, that every proved eruptive rock would contain a body of iron-ore. But the reasons why this sweeping conclusion is not at all warranted are natural and self-evident.

In the first place, as in the case of the furnace, all eruptives are not equally fluid. It might therefore happen that the weight of the iron grains would not be sufficient to cause them to settle through the viscous fluid. This alone would suffice to forbid the general expectation of an iron-ore body in every eruptive. Unequal fluidity of the mass as a whole might give rise, in one locality, to strings or bunches of ore wholly wanting in another.

Without pursuing further this obvious line of thought, we must consider the shape of individual masses of ore. Gathering in the manner that has been pointed out, the shape of the mass of ore could be nothing else than very irregular. Pure ore would be succeeded by pure rock and this in turn by lean ore and again rock. Nor, further, have we any clue as to how far in length, breadth or depth conditions favorable to segregation would obtain. With this cause dominant in the origin of iron-ores, obviously no safe estimate of the volume of reserve magnetites can be made. However great the outcrop, the extent of the ores depends upon the operation of conditions the extent of which there is absolutely no known method of determining. This much, however, can be predicted with reasonable certainty: the igneous iron-ores occur in undoubted igneous rocks which can be recognized almost unerringly. Moreover, the ores occurring in these rocks are, almost without exception, high in titanium.

All iron-ores high in titanium are exclusively in an eruptive rock generally called gabbro. Without exception, these ore-bodies are very irregular in outline. Every ore-body begins and ends abruptly. There is no leader, stratification, schistosity, bedding, or strings or lenses that will lead from one deposit to another.

There are many features peculiar to the titaniferous iron-ore deposits. Whatever the origin of the non-titaniferous iron-ores and their inclosing rocks, the gabbros are found overflowing or

overlying them, intruding in the form of sheets or cutting them in the form of bosses or knobs. The gabbro rocks and their titaniferous ores are younger than the non-titaniferous rocks.

Moreover, in the Adirondack region of New York, where iron-ores high in titanium occur in close juxtaposition with others practically free from this element, the titaniferous ores lie exclusively in the igneous gabbro, while the ores lying in the so-called gneisses are wholly unaffected by titanium. The most natural inference is that the two ores are wholly unrelated in origin, as to time, though both may be eruptive. Yet it is not certain that they may not be cotemporaneous. But, even in that case, they are sharply distinguished. The segregated titaniferous ores occur in an eruptive gabbro; the non-titaniferous ores in a distinct gneiss; the former as isolated pockets, sometimes of great size; the latter as extensive sheets, so regular and conformable to the schistosity of the inclosing rocks as to make prospecting, areally, for an extension of a given outcrop a safe business. Finally, the titaniferous iron-ores contain (in addition to titanium) chromium, vanadium, and nickel, from which the non-titaniferous ores are relatively free.

Hence, in a district where gabbro rocks are dominant, however extensive the showing of iron-ore in these rocks, no estimate of ore beyond three visible dimensions can be made; while in a district in which the dominant rock is the gray gneiss of known iron-mines, the possibilities of magnetic iron-ore reserves are bounded only by the limits of the gneisses. Of course this rule can be applied only in accordance with sound judgment based upon adequate experience. The titaniferous ores are undoubtedly of eruptive origin. Whether the non-titaniferous ores are primary or secondary, the final limits of a given horizon or stratum have never yet been found.

In conclusion, there is one possible origin of the magnetic iron-ores which has not been generally considered.

In the province of Oriente, Cuba, there are deposits of iron-ore that literally cover square miles of territory, save for division by drainage-streams. One belt is, to my personal knowledge, more than 10 miles long by 6 miles wide. The deposit

occupies the surface and extends from the "grass roots" down to a depth varying from a few inches to more than 100 ft. It is supposed to have originated by a process the very reverse of commonly accepted methods. Here a rock, originally high in iron, has been leached of its refractory silica, leaving as a residue, alumina, and carrying from 20 to 50 per cent. of metallic iron. This ore is not confined to Oriente, but is found extensively in Camaguey, covering large areas, and more than 100 ft. thick.

This variety of iron-deposit is supposed to have been caused by the chemical activity of meteoric waters in a warm, moist climate.

But our world was, during one long period of its history, probably subject, over its whole area, to precisely these conditions. There is thus a possibility, not to say a probability, that many iron-ore deposits may have been formed in this way. All that Cuba would need, to convert into magnetite her deposits of soft ores, would be a mantle of soil and a later metamorphism; and it is not difficult to imagine similar conditions as having resulted in our extensive iron-bearing gneisses with their deposits of magnetic iron-ores.

Viewed from any standpoint of genetic theory, then, we can only come to the conclusion that our magnetic iron-ore reserves are very great, and that, in a given district, their extent can be almost, if not quite, as accurately estimated as in the case of the Clinton ores.

When we consider, further, the fact that along the Atlantic coast, for 2,400 miles, from Ungava Bay to Alabama, and for an average width exceeding 100 miles, not only are the iron-bearing gneisses known to occur, but in this area many outcrops actually do occur from 1 to 7 miles along the strike, there is as yet no known reason why a proportionate depth cannot be safely assumed.

Conclusion.

It is hardly necessary to say that the foregoing argument does not deal with the question of ore-reserves "in sight," or actually indicated by recognized outcrops only. It does not warrant sanguine estimates of the value of particular undeveloped properties. It simply seeks to show that we are justified

in assuming for our magnetic iron-ores a potential extent, in something like the way we calculate upon other iron-ores; and that, in this respect, the estimates of some geologists and the real or affected fears of others are alike groundless.

The San Nicolás Mining-District, San Nicolás, Tamaulipas, Mexico.

BY IRVING H. WENTWORTH, MATEHUALA, SAN LUIS POTOSI, MEXICO.

(New York Meeting, February, 1912.)

THE little town of San Nicolás, Tamaulipas, Mexico, lies approximately 60 miles east of Linares, situated in the State of Nuevo Leon, and 150 miles SE. of Monterey, as shown in the sketch-map, Fig. 1. The place is reached from Monterey by the Tampico train as far as Linares, a distance of 92 miles, and then by coach the rest of the way. It is situated in a narrow valley between low hills, the houses forming one main street and a plaza. It is peopled by about 500 inhabitants.

The climate is temperate, the thermometer ranging from 90° in summer to a few degrees below freezing-point in winter; but this severe weather only lasts a day or two at a time, when the "northers" blow.

There are no streams in the vicinity, but water sufficient for the use of the people is obtained from wells. The rainfall is about the same as that in the rest of NE. Mexico. The rainy months are July, August, and September. The rest of the year is practically dry, with an occasional showery day or two during the winter, when a "norther" is blowing.

Timber is abundant in many localities. There are two varieties of oak, besides mesquite, ebony, "maguira," "patol," and a number of other kinds of trees indigenous to this region.

I. HISTORY.

On the discovery of the silver-lead bearing veins occurring in this district, the mining-camp of San Nicolás was founded in 1767 by the Spaniards, and numerous mines were opened and worked for a distance of about 3 miles east and 5 miles west of the town. At the western extremity and near the Carroleña mine, a small town called San Pedro was built. The

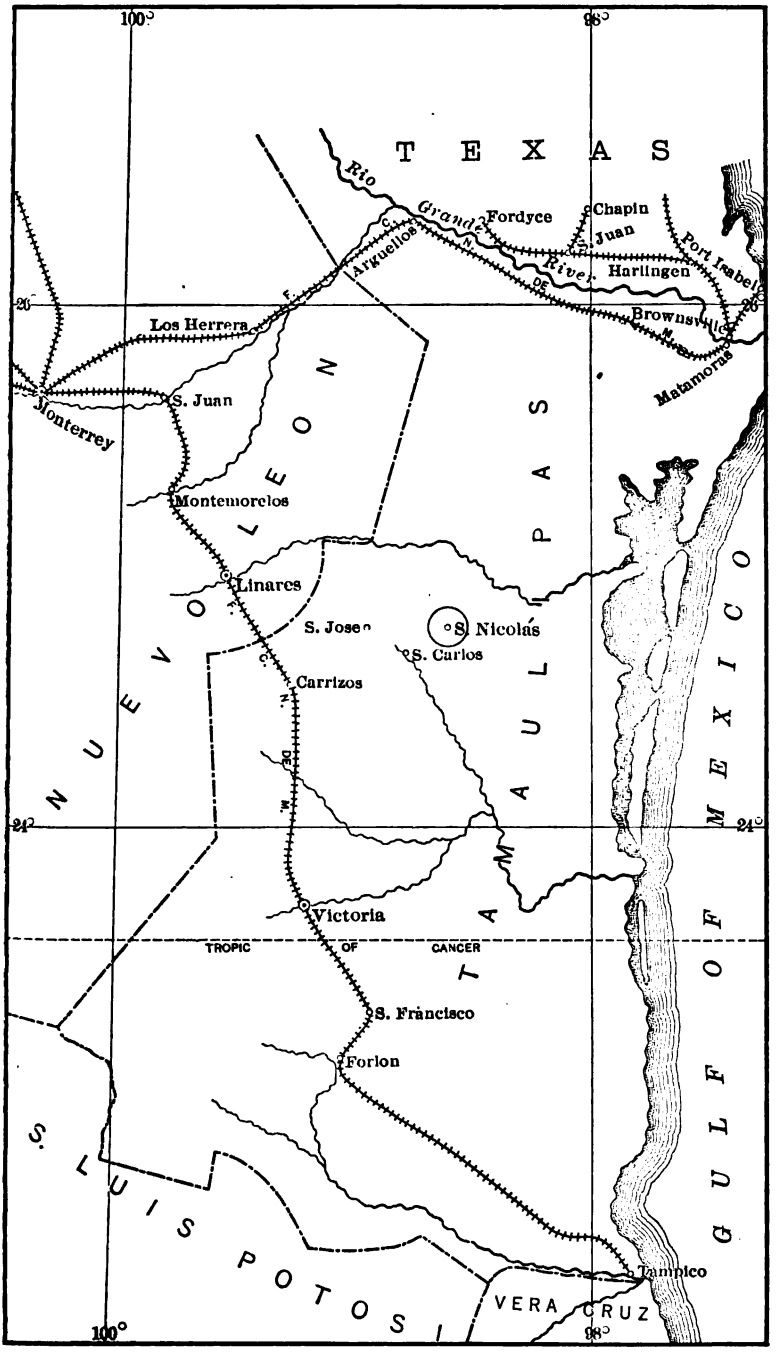

FIG. 1.—SKETCH-MAP OF THE STATE OF TAMAULIPAS, MEXICO.

ruined walls of the houses form a single street, and all is overgrown with brush and small trees. A little farther east of this and near the Santa Bárbara mine are the ruins of another settlement. Judging from the old records preserved in the archives of San Nicolás, together with the innumerable mine-openings scattered along a line 8 miles long, great activity was displayed in those early days, and the population grew to 14,000 inhabitants. All was suddenly brought to a stop in 1810, when war broke out with Spain, resulting in the independence of the country. The political disturbances which followed in rapid succession up to about 35 years ago, together with the fact that the country was over-run by wandering tribes of hostile Indians from the north, prevented further development of the camp.

The system of mining employed by the old Spaniards here was the same as that carried out in many other parts of the country, viz.: small irregular openings reaching to no great depth. The ores were treated by smelting in small furnaces, sometimes at the mines, and sometimes at some distance away. These sites are marked by piles of slag, some of which are to be seen as far off as the town of San Carlos, 18 miles SW. from San Nicolás.

From all appearances, the location of these *galemes*, as they are called, depended on the supply of water, or wood, or both together; for a particular kind of wood was used, called *ocotillo*. It is a small tree that grows throughout this part of the country. The wood is hard and burns with a fierce heat.

On the opening of the Monterey & Mexican Gulf railroad (now a part of the Mexican National system) through to Linares at the close of 1889, much needed means of communication with the Monterey smelters was provided. Mining-operations were renewed by progressive individuals in some of the old properties, and new ones were opened. Further facilities were afforded to the district by the building of the Linares & Gulf railroad by the San Carlos Copper Co. from Linares to San José in 1906, which shortened the haulage in carts from 60 to 25 miles. The traffic on this road, however, was stopped in January, 1911, by the closing down of the company and the sale of its premises.

II. Geology.

In traveling from Linares to San José, one crosses first a broad rolling country well timbered with mesquite, and beneath the alluvial soil lie slates and shales of Jurassic age. After covering two-thirds of the distance and on approaching the foothills of the San Carlos mountains, one comes to Tertiary conglomerates, the débris of the Cretaceous formation, which soon follows. This latter covers the entire district that is the subject of the present article, and may be divided into three distinct parts, each with marked characteristics, and all lying conformable one with another. The lower part is composed of approximately 200 m. of thick-bedded gray limestone containing 94 per cent. of $CaCO_3$, the middle part of 50 m. of a thin-bedded siliceous limestone from 4 to 12 in. thick, streaked with bands of chert, and the upper part of slates, impure limestone, and shales, about 200 m. thick. One class of rock changes abruptly to another without any slow gradation.

The San Carlos mountains form a rugged ridge extending south from San José for a distance of 12 or 15 miles, and rise to a height of 4,000 ft., more or less, above the plain, or about 5,000 feet above sea-level. Fifty miles east, there is a similar ridge running parallel to the first; and in between the two, and extending to the north, the limestone, with a breadth of 20 miles, rises to an average elevation of 2,500 ft. above the sea-level.

This is deeply furrowed by canyons over the whole extent, with drainage N–S., in which directions it slopes gradually from the center. Crossing this formation are a few wide folds with gently-sloping sides, whose axes run N–S. Besides these, there are other folds running E–W., which are short and sharp, forming monoclines and synclines. It is along the latter that the limestone has been fractured, and mineral-bearing veins have formed.

The country has undergone two different periods of igneous intrusions during Tertiary times. In the first, large bodies of syenite penetrated the limestone at San José and to the south, altering the surrounding rock in places to marble, and also forming garnet and other minerals peculiar to contact-metamorphism. It is along these contacts, and especially in close relation to the garnet, that the copper-ores in the mines at San

José are found.[1] In the second period, a large number of dikes and laccoliths of basalt were formed throughout this part of the country. The limestone, which has just been described, lying between the two ridges of mountains, was elevated during this epoch, forming a broad dome. Below this dome, there undoubtedly exists a large laccolith of basalt, for the center is pierced with a large mass which forms the Sacramento mountain, a single peak with precipitous sides, rising 500 ft. above the limestone. This peak is situated 3 miles west of San Nicolás.

Numerous dikes and small laccoliths are scattered over the entire limestone-area, having their source probably in the main laccolith. These intrusions have altered the limestone but little, but indirectly they have formed the means of the mineralization of the district.

We have thus a limestone-district folded and faulted by igneous intrusions; and the fractures resulting from these movements became the water-courses through which the warm solutions in their upward flow have formed metalliferous veins.

The principal mineral is argentiferous galena, with a little lead carbonate and sulphate occurring generally as scales around the sulphide where the latter has been oxidized. Occasionally, small amounts of blende occur associated with the galena. The gangue-matter of the veins is barite, calcite, and in some mines quartz. Often the ore is imbedded in a hardened clay derived from the adjacent rock by faulting. In the upper part of the veins, silver chlorides often occur in minute particles disseminated through the gangue; and while in most cases the amount is insufficient to be of economic interest, owing to present conditions, still paying values are often encountered.

The bulk of the ore recently mined in the district has been taken from the siliceous limestone, which forms the center of this formation. The mineralization has been effected by replacement of the limestone by the minerals brought up from below by warm solutions. These waters have been arrested in

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[1] J. F. Kemp, The Copper-Deposits at San José, Tamaulipas, Mexico, *Trans.*, xxxvi., 178 to 203 (1905). G. I. Finlay, Geology of the San José District, Tamaulipas, Mex., *Annals of the New York Academy of Sciences*, vol. xiv., pp. 247 to 295 (1901–03).

their upward course in a number of localities by impervious or semi-impervious strata, and the solutions have penetrated along the bedding-planes of the limestone just under the slate, forming blanket-veins, some of which are 15 m. wide. This restriction in the upward circulation of the water has caused the ore to have a great tendency to follow certain strata; so that instead of the vertical or oblique ore-shoots, such as are predominant in nearly every mine, there are horizontal ore-bodies. One characteristic of these ore-bodies, and a favorable one at that, is their regularity and extension. On the Todos Santos vein, these shoots have been followed for 1,000 m. with the ore still continuing; and in the Moctezuma mine, the vein has been explored 500 m. without finding any change. The veins penetrating the slate that composes the upper part of the formation are narrow, and the ore occurring in this rock is pockety. The narrowness is due to the fact that the slate possesses more resistance than the limestone; so that the fractures narrow down immediately after passing above the siliceous limestone, become ill-defined, and then, in most places, disappear before the surface is reached. In the immediate neighborhood of San Nicolás and in this slate, the galena occurs in quartz, which in many places forms bands along the walls of the veins, while the center is filled with calcite which is practically barren. Farther west, however, at El Palmar and Moctezuma, the veins in the slates consist almost entirely of calcite, carrying galena and silver chlorides. In the extreme western part of the district no quartz is found.

III. Veins.

The Todos Santos Vein.—This vein passes through the Mariposa, Todos Santos, Cuatro Vetas, La Condesa, and El Conde claims, which are located in the extreme western part of the district, as may be seen in the accompanying map. Its course is 70° NW–SE. astronomical, and follows a fold in the rock that in some places forms a syncline, while in others a monocline, see Fig. 2. Most of the ore occurs on the south side of the trough, in some places dipping north, but the general dip is to the south at an angle of from 70° to 80°. The ore in the vein varies from a few inches to 5 or 6 ft. in width, averaging about 2 ft. The blanket-veins, or *mantos*, as they are called

here, are composed of almost pure galena, varying from 1 to 8 or 10 in. wide, but 5 or 6 in. is about the average width. At the junction of the vertical vein and the *mantos*, the ore in most cases widens out considerably. The blanket-veins are always well defined, but the vertical vein has no regular walls, the ore penetrating the limestone at variable distances, with a tendency to form small *mantos* along the bedding-planes.

The workings on this vein have uncovered two *mantos*, the principal one being just under the slate, and the other one about 15 m. below the first.

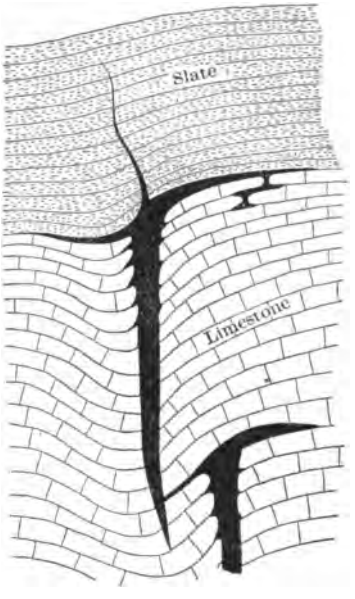


FIG. 2.—TODOS SANTOS VEIN. CROSS-SECTION.

An anticline passes through the center of the Todos Santos claim with its axis running N–S., the formation dipping westward through the Mariposa and beyond for a distance of 8 or 10 km., and eastward about 6 km. to El Palmar. An *arroyo* has cut down the top of this anticline, exposing the thick-bedded gray limestone. The dip of the formation both east and west is gradual; and although in places the angle is as much as 15°, the total dip to the east at the end of the 6 km. is about 100 or 125 meters.

The vein penetrates the slate above the siliceous limestone but a short distance, and consequently only shows on the sur-

face near the Todos Santos anticline. On account of the difficulty of locating the vein, nothing is known of it beyond the workings on the claims above mentioned.

The Moctezuma Vein.—This vein lies about 500 m. south of the Todos Santos vein, and is parallel to it. It passes through three claims, the Andrea, the Moctezuma, and the San Andrés; and the camp is known as Moctezuma.

It is a stronger vein than the Todos Santos, as it penetrates the slate to a greater distance above the siliceous limestone; but the *mantos* that have formed on it are small and irregular. Still it is to be noted that just under the slate the ore is wider and is of a better grade for some meters in depth than farther down, thus forming a horizontal shoot.

The dip of the formation here is to the east, the same as along the Todos Santos vein. Owing to the difficulties in prospecting here, nothing is known of this vein either east or west of the camp.

The Carroleña Vein.—The next known vein is the Carroleña, lying between 400 and 500 m. south of the Moctezuma, and running about 73° NW–SE. It passes through the Santa Claus, Maria, and Carroleña claims, and then turns S. 53° E. into the Dulcinea; thence east into the Don Quijote and San Nicolás, where it makes another turn to S. 35° E., and passes through the Santa Bárbara and Desamparados.

The Desconocida Vein.—This vein is parallel to the Carroleña, and lies about 75 m. to the south. It traverses the Desconocida, Dulcinea, Don Quijote, and the rest of the claims crossed by the first vein.

The Hormigas Vein.—About 100 m. south of the Desconocida is the Hormigas vein, passing through the Dulcinea, Hormigas, and Olivares claims.

Taking the three veins together, they are similar to the Moctezuma, being strong fractures, penetrating far up into the slate, and can be seen on the surface in many places. To the east on the Santa Bárbara and Desamparados claims, they combine the features of the Todos Santos vein by forming *mantos* just under the slate. Their distorted course is due to local horizontal faulting. The mines located on this group of claims form the camp known as El Palmar.

As has been mentioned before, the formation dips east from Todos Santos to El Palmar; thence it rises towards the east to a point just beyond the Desamparados, where the lower limestone shows again on the surface.

Still further, the country dips again slightly to the east on past the town of San Nicolás.

Beyond the Desamparados are a number of claims strung out to the east, as shown in Fig. 3. These claims are located along three distinct parallel veins, which outcrop through the slate. It is chiefly along these outcrops that the old Spaniards worked. There is a probability that these three veins are the continuations of the El Palmar group. The dip of all these veins is more or less the same as that of the Todos Santos.

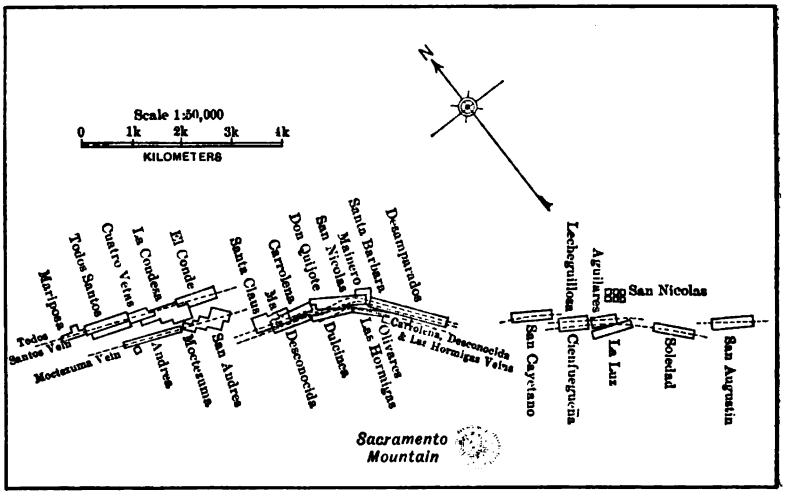


FIG. 3.—MAP OF SAN NICOLÁS MINING-DISTRICT, TAMAULIPAS, MEXICO.

IV. ORE-TREATMENT.

Since the renewal of mining-operations in this district, the ores have been treated by hand-concentration. First, the clean lump galena is picked out of the ore as it comes from the mine, and the rest is then passed over a 1-in. or a 0.75-in. mesh screen. The coarse dirt is ground in Chilean mills, gyratory, or jaw crushers, and, together with the fines, is washed on *planillas* to get rid of the clay. At the same time, the bulk is reduced more or less to two-thirds by throwing away the light

and worthless dirt that is carried down to the bottom of the *planillas*. After that, the ore is jigged by hand in a box from 3.5 to 4 ft. long by 15 or 18 in. wide, with a 4-mesh screen in the bottom, submerged in one end of a box about 5 ft. square, which contains the water. The jerking motion is produced by means of a pole from 12 to 15 ft. long.

The large box is divided in the middle by a baffle-board that reaches nearly to the surface of the water. Most of the ore fine enough to pass through the 4-mesh screen collects in the compartment under the jig-box, while the motion of the jig throws the light and worthless slimes over the baffle-board into the next compartment, where they collect.

The ore thus treated gives a product that assays as follows: Silver, from 1,500 to 3,500 g.; lead, from 60 to 70; insoluble, from 8 to 12; iron, 0.5; lime, from 2 to 3; and sulphur, from 5 to 8 per cent.

The camps that have been producing during the last 20 years are the Todos Santos, Moctezuma, and El Palmar. The other mines farther east have been operated but little. On all the properties, the workings are comparatively shallow, the shafts ranging from 40 to 125 m. deep.

The lower limestone has never been explored by any of the companies, owing principally to the depth necessary to undertake the work.

The miners have contented themselves in working along under the slate at small depths, although in this lower limestone there is a probability of encountering large deposits of ore along the fractures.

A New Electric Miners' Lamp.

BY D. B. RUSHMORE, SCHENECTADY, N. Y.

(New York Meeting, February, 1912.)

TORCHES were used by the early Romans for mine-lighting, and these were followed by open lamps or earthen jars filled with tallow or oil, and later by candles. In early coal-mining, explosive gases seldom occurred, and, if they were encountered, the danger of explosion was materially diminished by the absence of any ventilating-system and the consequent insufficient circulation of air to form with the methane an explosive mixture.

As the art advanced, the danger of fire-damp became more apparent, and methods for testing the air were resorted to as early as in the fifteenth century. The method of testing consisted in lowering a dog in a basket down the shaft. As soon as he encountered fire-damp he would commence to howl, after which he was withdrawn, and a leafy bush was fastened to the end of the rope and run rapidly up and down the shaft. This so disturbed the accumulation of gas as to cause it to rise out of the shaft. Later this odd method of testing was improved upon through the discovery that a light inserted in a body of fire-damp would produce a blue flame. Candles were used for this purpose, and, when fire-damp was thus discovered, brushing or some other primitive method of fanning was resorted to for driving out the gas. In some instances the gas was burned away, regardless of the danger thereby incurred. A man, wearing a well-moistened coat and a mask with glass spectacles, descended in the mine a few hours before each shift. With a long stick, to which was fastened a burning torch or candle, he would crawl on the floor with the stick so raised that the light would sweep the roof and ignite the gas which it encountered. In this position the flame of the explosion would pass over him, after which he would retreat from the gases resulting from the combustion. The occurrence of serious accidents resulted in

the substitution of a method by which the candle was moved along the headings by means of an endless rope, which was supported on pulleys near the roof and could be operated from a safe point.

In 1815 Davy invented the safety-lamp which bears his name and the principle of which is embodied in the numerous safety-lamps of later inventors. Although many improvements have been made in the safety-lamp, it is still far from perfect; and the present indication is, that, in order to obtain a perfectly safe and reliable lamp, the electric light must be resorted to.

In metal-mines, and even in coal-mines where the gases contained in the atmosphere are sufficiently diluted to render them harmless, illumination may be had by any form of naked light. But in collieries where considerable quantities of fire-damp and coal-dust are liable to collect, the lamp-flame must be protected from direct contact with the gas, and it becomes necessary to use some sort of safety-lamp.

The most common of all miners' lamps is the small tin lamp which generally is carried on the miner's cap. A cotton wick extends through the spout into the fluid, which is generally some kind of petroleum-product. While this lamp, when trimmed and kept in order, gives a fairly-good light, the objections to its use are many. The almost continuous smoking seriously impairs the purity of the air and thus handicaps the miner in his work. It is easily blown out by blasts and drafts. The leakage of oil from the lamp is a constant annoyance, and the time required for refilling and trimming the lamp is considerable.

The acetylene lamp, used to a considerable extent in a number of mines, is a marked improvement on the oil-lamp. Not only is the illumination materially better, but the smoking and leakage of the oil-lamp are practically eliminated. The difficulties of handling seem, however, to have prevented its general adoption.

In mines liable to the intrusion of excessive amounts of fire-damp, safety-lamps must be used, either exclusively, or for purposes of inspection, preliminary to the removal of the gas. Several types of safety-lamps of more or less ingenious design have been placed on the market and are in actual use in the various gaseous coal-mines.

The Davy is an ordinary oil-lamp, the flame of which is surrounded by a cylinder of wire gauze, with a top of the same material. The principle of this lamp is that the cooling effect of the wire gauze prevents the propagation of the flame from the inside of the lamp to the outside atmosphere. After the lamp has been filled with oil and lighted, it is locked, so as to prevent the miner from having access to the flame, the wick of which is trimmed by a wire passing up through a close-fitting tube from the bottom.

The Clanny lamp is similar to the Davy, with the exception that the lower portion of the wire-gauze cylinder is replaced by a short glass cylinder, giving a somewhat better illumination.

Stephenson's lamp has a long cylinder of glass surrounded by wire gauze, and bonneted above by perforated copper. The air-feed is also through the gauze, going underneath and into the cylinder to the flame, thence out of the top as usual. This keeps both cylinder and gauze cool, and its relative security rests essentially on the regularity of the draft, for if the inside air becomes overheated the light goes out; so it must be suspended properly.

The Marsaut lamp is an improvement upon the Stephenson type, and can safely stand a fair amount of tilting. A great difficulty is, however, experienced in relighting it, and from the winding path pursued by the feed-air, proper circulation does not take place until the lamp gets hot.

The Hepplewite-Gray lamp admits the air at the top, down four tubes, and through an annular chamber above the oil-vessel. The only gauze employed is that covering the outlet and the annular inner chamber. A serious difficulty with this lamp is its liability to be extinguished when suddenly lowered.

In the Dick port-hole lamp, the air enters the lamp above the case, passes through the gauze, then descends to the flame, while the products of combustion rise inside the lamp, to be emitted through circular holes at the top of the bonnet. The bonnet is made of a seamless steel tube, and is light and strong.

The Wolf benzine safety-lamp[1] is a departure from the above safety-lamps in that: (1) it burns benzine or naphtha; (2) it contains a self-igniter which permits the relighting of the lamp

[1] *Trans.*, xiii., 129 to 132 (1884–85); xiv., 410 to 418 (1885–86).

50 times without opening; and (3) it contains a locking-device which cannot be opened except by the use of an exceedingly powerful magnet.

While the safety-lamp possesses great value as a gas-detector, it is open to several serious objections. The first is the possibility that the gas, entering the lamp, and burning inside the cylinder, may overheat the gauze, which will then permit the flame to pass through it. Several types of lamps, however, are arranged to be self-extinguishing in case of ignition of gas inside the gauze. This is, of course, objectionable, as it will leave the men in darkness when such a danger occurs. The temptation of the miners to relight extinguished lamps, or open them for lighting pipes, etc., is very great, and has in many instances resulted in serious explosions. Finally, the illuminating-power of the lamps is very low.

For the above reasons safety-lamps of the ordinary type are being replaced in many localities, especially in bituminous coal-mines, by portable incandescent electric lamps. This type offers numerous advantages: increased safety in explosive atmospheres; increased illumination, ability to concentrate the light on a certain area, reliability and simplicity, elimination of smoke, cleanliness, smaller fire-risk, etc.

An objection often raised against the electric lamp is that it offers no means of indicating the presence of fire-damp. Several devices have been proposed, such as providing spiral wires coated with salts of platinum, etc.; but there seems to be at present no really satisfactory solution of this difficulty, although there is no reason why a solution should not be expected. To attempt to find a satisfactory fire-damp indicator is, however, merely to delay the general introduction of electric lamps; and there seems to be no real necessity for such an addition to a miners' lamp. The fire-damp detector might well be an entirely separate instrument, which could be installed in places where the fire-damp is expected to form first, such as near the roof, or in cavities therein. There is no reason why one or more gauze-lamps should not be supplied with each batch of electric lamps, and hung in the working-place where they can be easily seen, and where any outburst of gas would be most likely to occur.

Numerous electric mining-lamps have been designed, but few

have proved efficient and durable. The design in the United States and abroad has followed very different lines. In Europe the hand-lantern has been almost exclusively considered, while the tendency here has been towards the perfecting of a combined hand-lantern and head-light. This is the logical result of the fact that the most satisfactory working-results can be obtained by a head-light, whose reflector concentrates and throws the light in the direction in which the miner works.

A new electric miners' lamp of improved design has recently been developed by the General Electric Co. to meet the increasing demand for a safe and reliable lamp. The idea of a systematic investigation as to the possibilities of developing such a lamp originated with W. J. Richards, Vice-President and General Manager of the Philadelphia & Reading Coal & Iron Co., to whom credit is due for his very active assistance.

The lamp is of the combination hand-lantern and head-light type, and consists of a miniature tungsten lamp-unit, operated from a light storage-battery. The lamp is provided with an efficient reflector, designed to illuminate a 9-ft. circle at a distance of 4 ft. from the lamp, and photometric records show about 2.5 or 3 candle-power at a distance of 2 ft. The battery has a capacity of 5 ampere-hours, and is of sufficient size to operate a lamp from 12 to 14 hours.

When used as a head-light, the lamp, with its reflector, is fastened to the miner's cap, and connected by means of a flexible cord to the battery, which is carried on a belt or from shoulder-straps, as shown in Fig. 1. When used as a hand-lantern, Fig. 2, the lamp-socket is removed from the cap-receptacle and inserted into the receptacle on the side of the battery, simply taking the place of the cable-attaching plug.

Some of the important points in connection with this design are: substantial construction; light weight and compactness; complete inclosure of all conducting parts; use of an absorbent for the electrolyte, to prevent spilling; small number of movable contacts; increased efficiency of a 4-volt lamp; efficiency of the reflector; impossibility of charging with reversed polarity; instant convertibility from hat-lamp to hand-lantern. These features will be considered in the order in which they have been mentioned.

Special care has been taken to select all materials used in

FIG. 1.—ELECTRIC MINERS' LAMP USED AS A HEAD-LIGHT.

FIG. 2.—ELECTRIC MINERS' LAMP USED AS A HAND-LANTERN.

the construction to withstand the hard service to which this lamp may be subjected. The steel battery-case is drawn from a single plate and is, therefore, seamless. The battery-jar is not of the brittle rubber commonly used, but has some elasticity left in it, so that, should the steel case receive a blow, there is little chance of the battery-jar being cracked. The cord from the battery to the lamp is metal-armored where the greatest wear and strain occur, but perfectly flexible where flexibility is required. The battery-case is locked with a magnetic lock which also locks the attaching-plug in the receptacle, thus preventing this plug from working loose and permitting the formation of a spark. The hat-lamp is sealed with a lead seal, which must be broken before the miniature lamp-globe can be removed.

This outfit has been designed, as already explained, for use as a head- as well as a hand-light, and therefore the weight of the battery, which must be carried on a belt, in the pocket, or by shoulder-straps, has been kept at a minimum consistent with strength and durability. The battery is flat and compact in order that it may lie as close as possible to the body. It weighs approximately 3 lb. and measures $6\frac{1}{2}$ by $4\frac{5}{8}$ by $1\frac{1}{8}$ inches.

In order to prevent accidental short-circuits, causing sparks, all live parts are totally inclosed in molded compound. The battery-terminals, attaching-plug, hat-receptacle, and lamp-holder are all molded from hard fire- and acid-proof compound and thus perfectly protected from corrosion due to acid spray or gases.

An absorbent is used for separating the battery-plates and also filling up the extra space in the battery, except the mud-space. This absorbent acts like a sponge and holds the electrolyte in the battery even though it be inverted for some time.

Where low voltages are used, the pressure-drop caused by the contact-resistance may be serious. The movable contacts in this lamp have, therefore, been reduced to a minimum and have been so designed that they come together with considerable pressure.

The increased efficiency and life of the apparatus is principally due to the improved tungsten lamp. A 2-volt lamp is somewhat inefficient and has a comparatively short life. This

is due to the cooling effect of the large leading-in wires on the short filament. This cooling by conduction reduces the temperature of about one-third to one-half of the filament to a point where little or no light is produced. In order to produce the rated candle-power, the rest of the filament must be operating at a temperature far above a safe value, with a consequent unreliability and shortening of life. The filament of the 4-volt lamp, for which this outfit is designed, is about twice as long as a 2-volt filament, so that the percentage of length cooled by the leading-in wires in a 4-volt lamp is about one-half that in a 2-volt lamp. This allows the rest of the filament to be operated at a safe temperature, giving a reliable lamp of long life. The filament is, however, not so long as to cause mechanical weakness.

The reflector is made of stamped steel, enameled both inside and outside, thus absolutely protecting it from corrosion. Tests have shown that white porcelain enamel forms one of the best possible reflectors, and is very durable and easy to keep clean. The lamp-globe is protected from injury by a glass plate, fastened to the rim of the reflector by a spring. This glass further tends to prevent dust from accumulating on the inside of the reflector.

The contacts on the battery-terminal and plug are made concentric, so that, no matter how the plug is put in, the polarity must be the same each time. A separate plug is supplied for use in charging, and can be wired permanently on the charging-rack with its polarity correct. It will then be impossible to charge the battery from this plug with reversed polarity.

While the demand for an electric mine-lamp seems to be almost entirely for one to be worn on the cap, to supersede the present oil-lamp, it appeared very desirable to have an outfit that could quickly and readily be converted into a hand-lantern. This has been accomplished by designing the lamp so that, by a quarter-turn, it can be removed from the hat-receptacle and inserted, by a quarter-turn, into the plug-receptacle on the side of the battery.

A Concise Method of Showing Ore-Reserves.

BY N. H. EMMONS,* KNOXVILLE, TENN.

(New York Meeting, February, 1912.)

THE work of a consulting engineer or manager, when controlling mining-operations, requires that he have all the information concerning the mine in as concise a form as possible, and as the ore-reserves and their depletion and enlargement are the vital points to be studied, it is well to have the records kept up to date and in some convenient form.

The present paper describes a method adopted by me, which shows in a very clear manner the ore-reserves in a mine. The case illustrated is that of a vein near enough to the vertical for the vertical elevation of the mine to be the main working-map. A flat ore-body would be represented on a similar plat, but instead of the horizontal thickness of the vein being shown in the circles, the vertical thickness would be inserted.

Fig. 1 represents an ideal longitudinal elevation of a mine, plotted in the ordinary way, using 100-ft. blocks and section-lines. In a mine where the block-numbers are posted by the engineer underground, the system of keeping the ore-tonnages extracted from the different blocks is easily maintained, and the actual tonnages of ore extracted can be kept so that the surveyor can have a check on his estimate from time to time.

The ore-reserve sheet, Fig. 2, should be the same size as the map it is to accompany. Each square is used for the description of the ore-reserve in the block it corresponds to on the elevation. Where the section-lines cross the levels a circle is drawn, in which are written the widths of ore at that point, these widths being used in the calculations of each block cornering in the circle. It has been found convenient to show the figures in black in the lower half of the circle when the actual widths of ore are known, and when it is necessary to make up the figures of probable ore, as at the time of making

* Consulting Engineer.

annual or semi-annual reports, red figures are written in the upper half of the circle to represent the probable thickness of the ore; and later on, when the development warrants it, the black figures are inserted underneath. The same color-scheme may be used for ore actually developed and probable ore, red

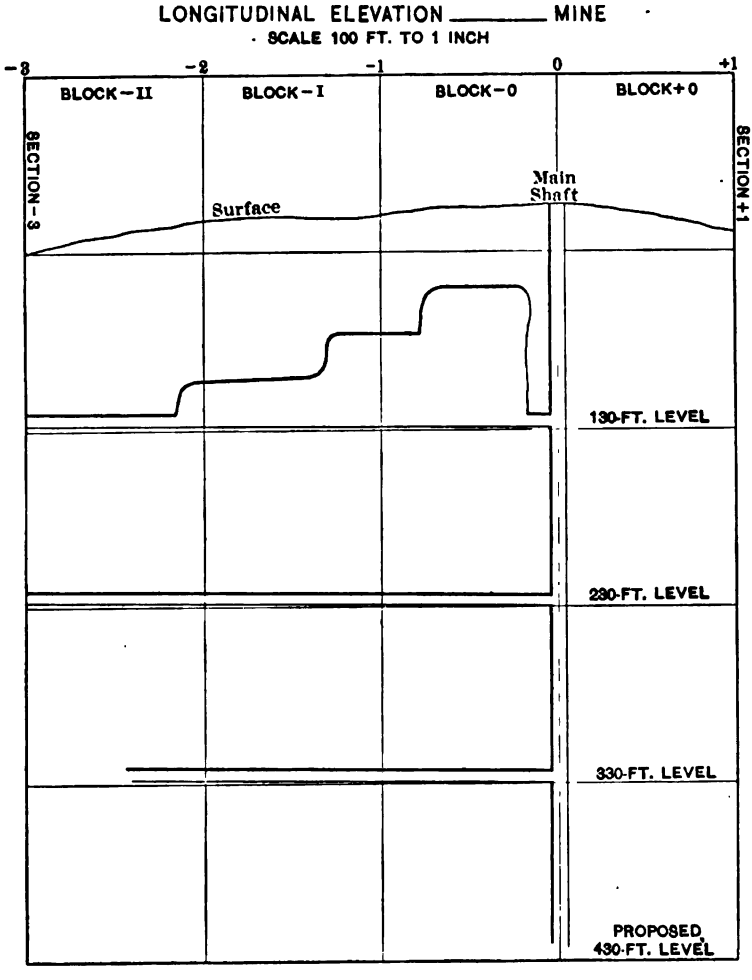


FIG. 1.—IDEAL LONGITUDINAL ELEVATION OF A MINE.

figures representing the probable ore being first inserted in the column headed "Developed," and when the ore is developed the black figures should be inserted in the time-space when the development was completed.

There should be an ore-reserve sheet for each year, and the space between the levels divided into four spaces if quarterly estimates are made, as shown on the 130-ft. level, Fig. 2, or into 12 spaces if the figures are worked up monthly, as shown on the 230-ft. level. Then there should be a sheet kept to enter the ore developed and extracted each year, 10 spaces being a con-

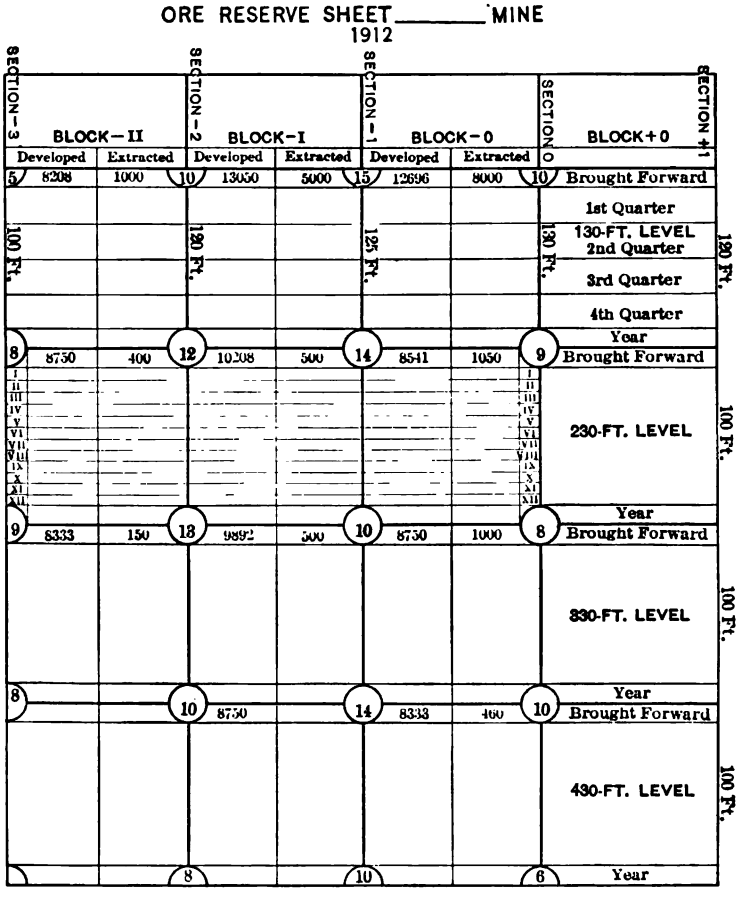


FIG. 2.—ORE-RESERVE SHEET TO ACCOMPANY ELEVATION, FIG. 1.

venient number, thereby covering 10 years, so that the record of the development and extraction by years can be seen at a glance.

In mines where the value of the ore is kept it is convenient to insert in one lower quarter of the circles the width of the

ore, and in the second lower quarter the value of the ore. The idea can be carried a little further by using squares instead of circles at the corners, in which the widths and values are written, and making a circle in the center of each large block in which to insert the average value of the block of ore.

As it is always well to have the figuring in some permanent form so that it can be readily checked, a sheet, Fig. 3, should be kept for each level, showing clearly how the tonnages were calculated.

ORE-RESERVE CALCULATION-SHEET ——MINE

First Level. January 1, 1912.

Block No.	Section No.	Average Width of Ore at Section.	Depth of Ore.	End Area or Area at Section.	Average End Areas.	Length of Block.	Volume Cubic Feet in Each Block.	Developed Ore, Tons. 12 Cu. Ft. — 1 Ton.	Probable Ore, Tons. 12 Cu. Ft. — 1 Ton.	Remarks.
	0	$\frac{10+9}{2}$	130′	1,235						All Ore Developed Previous to Jan. 1, 1912.
—0					$\frac{3,047}{2}$	100	152,350	12,696		
	—1	$\frac{15+14}{2}$	125′	1,812						
—I					$\frac{3,132}{2}$	100	156,600	13,050		
	—2	$\frac{10+12}{2}$	120	1,320						
—II					$\frac{1,970}{2}$	100	98,500	8,208		
	—3	$\frac{5+8}{2}$	100	650						

FIG. 3.—ORE-RESERVE SHEET.

If the ore-reserve sheet is posted monthly, a great deal of information can be gleaned, a splendid record of operations is obtained, and the work for the future is planned to the best advantage. Another important point is that the percentage of ore recovered, when a stope or block is worked out, can be readily seen, and estimates of the available ore in other blocks can be made with greater accuracy.

Each problem may require a different form of reserve-sheet, and the sheet can be varied to suit any mine-map.

One of the main advantages of the sheet is that it will teach the mining engineer to make careful estimates of ore-reserves, and not, as in many cases is done now, allow the mine-foreman to say that there are so many tons in such and such a stope that will require so many months to take out.

The development of the mine is so clearly shown, if the sheet is kept properly posted, that the management can tell at any time whether the development is running behind the extraction, or just what is going on. There is no reason why a mine should not have its ore-reserve book balanced every month in as careful a manner as are the accounts, if such a system as the above is kept, and at the end of the year it will save many disappointments at the small amount of new ore developed if the question has been kept continually before those in charge.

Another point that is so often handled in a dilatory way at mines is the method used for figuring an estimate, and the keeping of this information regarding widths of ores, values of ore, and estimates of ore-bodies on various slips of paper or in numerous note-books. If the reserve-sheet is being kept, there will be definite methods and exact figures needed at least once a month or a quarter, and as these figures are obtained if there is a place to put them, they will be entered on the final sheet as soon as obtained.

Take, for instance, a drift in ore of varying width and assay. If the method of estimating ore in use at the mine is to sample and assay every 10 ft., then the average width and assay of the 50 ft. on each side of the circle or corner is made up and inserted in the circle, the amount of ore in the four blocks contiguous to the circle worked out, and posted in the month when the actual figures are made up.

If ore-reserves are classified, then a separate sheet should be used for each class of ore. Many mines produce smelting-ore and concentrating- or milling-ore from the same block. A smelting-ore sheet should be kept and the width of this class of ore alone be entered on one sheet, and that of the concentrating-ore on another.

Another far-reaching point in favor of such a record is that it can be seen at a glance if the high-grade stopes are being pushed too rapidly and the low-grade being neglected.

It is the custom of many companies to have maps of the

mines sent in with annual reports, with the intention of keeping the directors posted regarding the development of their property. How much better it would be to send in an ore-reserve sheet as outlined above, that would convey the exact information about what was actually being done at the property, and be easily understood.

Progress in Roll-Crushing.

BY C. Q. PAYNE, NEW YORK, N. Y.

(New York Meeting, February, 1912.)

THE art of crushing ores and other materials by means of rolls is a comparatively recent one. While the first record of rolls using iron crushing-surfaces dates back to the year 1806, when they were employed in Cornwall, their principal development has taken place during the past 30 years.

To Stephen R. Krom belongs the credit of the pioneer in introducing the belted high-speed roll, which has had its origin and a marked development in this country. His notable contribution to the art was in the use of a single bed-plate or frame supporting the roll-shafts, and to which levers holding the movable roll-bearings were pivoted. He also made use of steel tension-rods to support the crushing-strains, and of hammered-steel tires for the crushing-surfaces. These changes brought the design of crushing-rolls to a high level.

Following closely thereon, about 1885, W. R. Eckart conceived the idea of the swivel or ball-and-socket support for the roll-shaft bearings. This is an excellent mechanical conception, especially for bearings held against a spring pressure, and while it may not be necessary for all types of rolls, yet it has been quite generally adopted by other roll-designers, and illustrates the refinement which roll-construction has now reached.

Other engineers, as, for example, Argall, Vezin, Roger, and Sturtevant, to mention only a few among many, have also given the closest attention to the various details of rolls, such as the frame, springs, bearings, and shafts, and have developed many novel and original designs. It has remained, however, for the boldness and originality of Mr. Edison to extend the field of crushing-rolls in two new directions, and to cause them to ex-

ercise new functions. In developing crushing-machinery for his Portland cement works, Mr. Edison constructed "giant" rolls, having a diameter of 6 ft. and a length of face of 7 ft. With these he was not only able to challenge the long-established position of the jaw- and the gyratory crushers as primary crushers, but even to leave them well in the rear. By means of projecting knobs on the roll-surfaces he utilized the stored energy of the revolving rolls, and was able to shatter masses of rock of so huge a size that they could not otherwise be made to enter the rolls, thus saving the expense of block-holing and sledging, which is usually charged to quarrying. This work has had successful applications in crushing limestone rock, and there remain also possibilities of the extension of this new use of rolls to rocks of a still harder nature.

In going to the other extreme of size-reduction, Mr. Edison has also utilized rolls in pulverizing cement-rock in his work. For this purpose, rolls with sectional, corrugated, chilled-iron shells 30 in. in diameter and 8-in. face, are provided with shafts 18 in. in diameter, and are forced together with a spring pressure of 100 tons. The feed-material, which has passed a 0.75-in. screen-opening, is thus reduced in a single operation to a size at which 94 per cent. passes 100-mesh screen, at the rate of 60 tons per hour. This use of rolls carries the principle of choke-crushing to so extreme a limit as to involve practically a new function. It would appear at first sight, however, that there is less profit-margin for rolls in pulverizing than in mass-reduction, and only a close comparison of the final products obtained and the respective costs per ton can determine the relative economy of rolls for pulverizing when compared with the tube-mill and other types of grinding-apparatus.

Rolls for Coarse Crushing.

Irrespective of the part which rolls may play in the future in their relation to the two extreme limits of size-reduction, there is no doubt that they have achieved for themselves a secure position in crushing products of intermediate sizes. This is partly due to their large capacity and low cost of operation. It is also due to the fact of their mechanical simplicity, which involves the principle of the toggle-lever in overcoming crushing-strains exerted by particles brought within the angle of nip of their

surfaces. Since their revolving masses also serve to absorb their own "peak-loads" when properly fed, a moderate and fairly uniform application of driving-power is able to accomplish a considerable amount of effective work in splitting and shattering rock-fragments.

Perhaps the most distinctive advantage of rolls is that their construction permits them to apply the principle of "arrested crushing" to a greater range of sizes than is possible with any other type of crushing-apparatus. The crushing-pressure exerted by the opposing roll-surfaces during the angle of nip is instantly released and ceases when the rock-fragments reach the horizontal diameter of the rolls, where the open space between them permits the material to be discharged. Roll-crushing thus permits most careful and accurate stage-reduction within a wide range of sizes, and avoids pulverizing and sliming an undue amount of the softer minerals of an ore, in crushing it to the size at which they will unlock sufficiently from the surrounding gangue to permit their concentration to take place. For those ores, therefore, which require concentration, the use of rolls in preparing them for jigs, shaking-tables, or magnetic separators has become almost the universal practice. This applies to many iron-, copper-, lead-, and zinc-ores. Gold- and certain silver-ores, both those which require concentration and those which do not, are in a class by themselves, since usually their values can be extracted without essential relation to the granular condition of the crushed product.

The modern tendency to reduce milling-costs by increasing the mill-capacity has demanded a greater duty from rolls than ever before, and in the larger mills rolls are now employed from 36 up to 54 in. in diameter, and from 15 to 28 in. width of face. Such rolls are used mainly for coarse crushing; that is, they take the product from the jaw- or the gyratory crusher, from 1.5 to 2.5 in. in size, and reduce it to about 0.5 in. These coarse or No. 1 rolls are then followed by other rolls set closer together for finer crushing, and possibly by others which re-crush certain middlings products obtained in the process of ore-treatment, or even tailings, dependent upon the nature of the ore and its association. Rolls of this general character require massive construction and excellent work-

manship. Rolled-steel tires can now be obtained up to 54 in. in diameter. Special hard steels, such as chrome- and manganese-steels, are also used for certain ores, either in the form of tires or of plates bolted to a central mandrel. In this way the life of the crushing-surfaces has been much prolonged.

Marked progress has thus been made within recent years in the field of coarse crushing by means of rolls, in response to the greater demands of modern mill-practice, and this progress has been largely brought about by increasing the dimensions of the rolls and adopting a more massive construction, as well as a better design, combined with a wider choice of steel adapted to different ore-requirements than has heretofore been available.

Rolls for Fine Granulation.

On the other hand, it must be admitted that up to the present time rolls designed for fine crushing, where the ore-requirements demand a maximum granulating effect and a minimum pulverizing or sliming effect upon the crushed product, have made little progress compared with rolls designed for coarse crushing. In fact, rolls, as heretofore designed, can hardly be said to have held their own; and since little assurance of their satisfactory operation can be had in connection with an ore which must be reduced to pass a 20- or 30-mesh screen while retaining the crushed material in a granular condition, rolls have been assailed on all sides by various types of ball-mills and other pulverizing-apparatus which claim to accomplish the function of granulating an ore successfully, but usually by means of some reduction in the time during which the pulverizing effect takes place. While there may be an overlapping territory at the limit of fine granulation where pulverizing-apparatus may be so adjusted as to perform the function of approximate granulation with sufficient success to make their use advisable, yet it is clear that a crushing-force exerted upon material placed between walls which do not touch when at their minimum distance apart, must produce a distinctly better granulated product than when it is crushed between walls which are brought into physical contact with a grinding-pressure.

With the presumption of advantage thus on the side of rolls, even down to the finest sizes, the fact remains that heretofore

rolls have proved unsatisfactory and inefficient, from lack of control over the granulating action as the roll-faces wear, and also from their small capacity.

In looking more closely into the cause of this inefficiency, it is evident that the effect of the irregular wear of the roll-faces becomes a more serious matter in fine crushing than in coarse crushing, for the reason that in the former, since the faces must be brought quite close together, the ratio of the sectional area due to irregular wear to the total areal opening between the rolls is greater than in the latter case. Hence any ridging, grooving, or corrugating of the roll-faces permits a considerable proportion of the particles in the feed-stream which enter the depressions to pass between the rolls without being crushed. This reduces the capacity of the rolls, measured by the amount of "under-size" or finished product obtained. Another difficulty arising from irregular wear of the roll-faces is due to the fact that when out of parallelism rolls tend to exert a certain component of the crushing-pressure at right angles to the diameter of the rolls, or in the direction of the axes of their shafts. This produces an end-thrust upon the roll-shafts, which, transmitted by means of collars to their bearings, causes them to heat and the shafts to cut, thus absorbing power wastefully, and still further reducing the crushing-efficiency as measured by the power consumed to operate the rolls in relation to the amount of finished product obtained.

Rolls used for fine crushing thus show a decreasing efficiency in proportion to the wear of their roll-faces until a point is reached where it becomes necessary to stop the crushing-operation, and to restore the faces by chipping, grinding, or machining them until their surfaces are parallel. This involves loss of time and of mill-capacity, besides expense and a wasteful use of the roll-shells.

While a certain amount of choke-crushing is usually advisable in fine crushing, in order to offset some loss of capacity, the best results can only be obtained, where the prime object is to granulate an ore, when the roll-faces are maintained parallel, and when the feed consists of sized material in order to avoid packing and pulverizing it in passing it between the rolls. It seems clear from the above considerations that further advance in the art of fine granulation by means of rolls can only be ex-

pected by means of certain refinements of function whereby the roll-faces can be maintained parallel while they undergo wear.

Frazee Rolls.

Having had the problem of fine granulation in mind for some time past in its relation to the treatment of certain classes of ores, I have been gratified to find in a recent design of rolls by J. S. Frazee, of Brooklyn, N. Y., that he has completely overcome the obstacles which have heretofore prevented the satisfactory operation of rolls when used for fine crushing, and when a granulated product is required.

Mr. Frazee has given me the privilege of presenting a brief account of his rolls to the Institute; and as I believe they are not generally known, I take pleasure in doing this, in the confident belief that they mark an important step in the advance of the art of roll-crushing.

Figs. 1 and 2 show a side and an end view of a pair of 18- by 12-in. rolls. It will be seen from Figs. 3 and 4, which show the same rolls in part sectional elevation and plan, that the bearings of the roll shafts are supported in side-frames. These are of cast-iron, without tension-rods, and are held together by stay-bolts and lock-nuts to permit the width between the frames to be slightly adjusted, if necessary, when the rolls are in place. This construction has certain advantages for fine crushing over the single bed-plate or frame. It is lighter, less expensive, and permits the rolls to be completely boxed in by means of a wooden housing. This is hoppered at the bottom, and is connected with an exhaust air-pipe, through which the fine dust is drawn by means of a fan, thus keeping the mill-space free from dust. The bearings are of the solid type, lined with babbitt, and each provided with dust-caps at each end. All four are movable in the guides provided for them in the frames. Two of the bearings are held against a spring pressure, and the other two against screw adjusting-bolts, for the purpose of regulating and controlling the space between the roll-faces. These adjusting-bolts are provided with sprocket-wheels and an endless link chain, which is kept taut by means of a small idler between them. By inserting a long-handled spanner in openings in either of the sprocket-wheels, the adjusting-screws can be moved in exact unison in varying the position of the roll-space. This feature

FIG. 1.—SIDE-VIEW OF 18- BY 12-IN. FRAZEE ROLLS.

FIG. 2.—END-VIEW OF 18- BY 12-IN. FRAZEE ROLLS.

has been found a very convenient one, as it permits close and accurate adjustments of the gap between the roll-faces to be made while the rolls are running.

One of the essential features of Mr. Frazee's design consists, as shown in the illustrations, in making one roll-shell longer

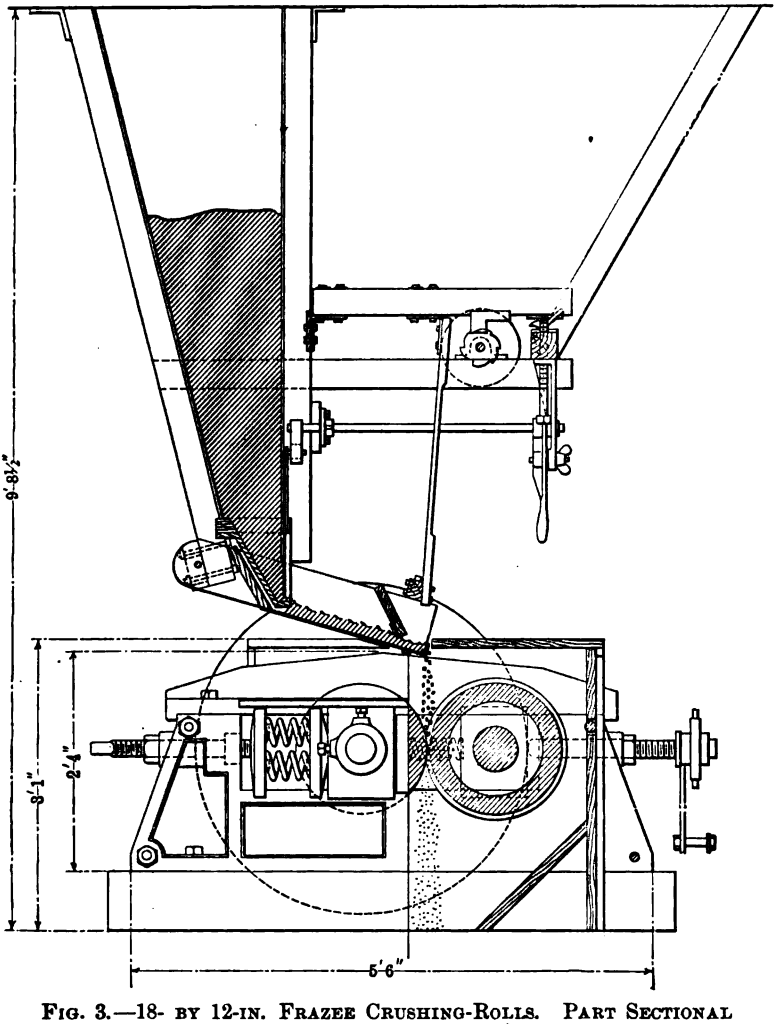


FIG. 3.—18- BY 12-IN. FRAZEE CRUSHING-ROLLS. PART SECTIONAL SIDE-ELEVATION ALONG LINES *D*, *E*, *F* OF FIG. 4.

than the other, so as to permit flanges to form on it, between which the shorter roll revolves, with a slight clearance. The rolls are fed from a hopper, which is supported independently of the roll-frame, and in such a position as not to interfere with

the removal of the rolls. The lower end of the hopper is made of cast-iron, and is provided with a gate, which is operated by means of a shaft and levers, so that it can be controlled from the forward end of the roll-frame. A feed-tray is placed below the hopper so as to convey the material to be crushed from the hopper-gate to the space between the rolls. This feed-tray is supported at the back of the hopper by a pivot-bolt, which allows it to rotate slightly, and at the front end by means of a wooden connecting-rod to a horizontal arm flexibly connected to a support at the front of the hopper. When the front end of the arm is raised by the teeth of the revolving cam below it,

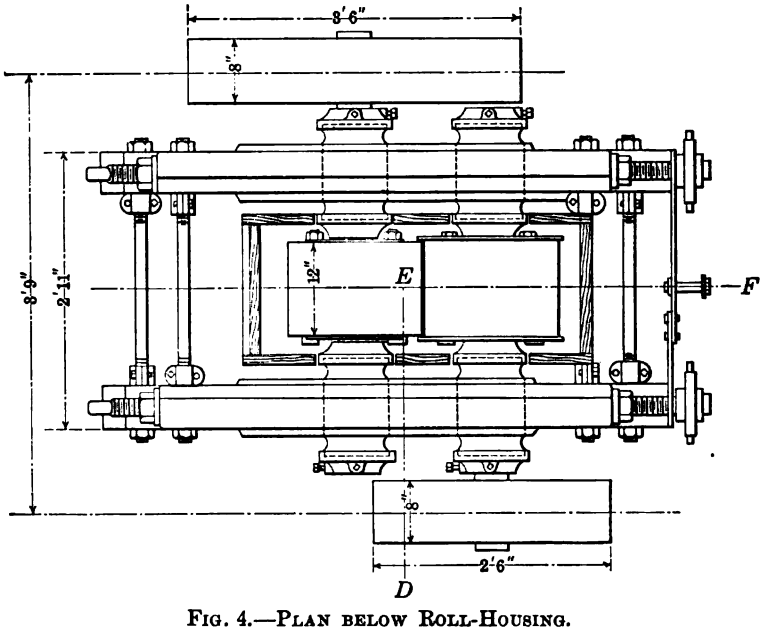


FIG. 4.—PLAN BELOW ROLL-HOUSING.

the discharge end of the feed-tray is caused to rise and fall with a jarring or bumping motion. This causes the material in the feed-tray to be carried forward and to be discharged in a steady stream over the angle-iron which forms a lip or dam at the extreme edge of the tray. The bottom of the feed-tray is made of sheet-steel to permit the material to slide freely. The amount of motion imparted to the end of the feed-tray can be varied by means of an adjusting screw-bolt, which regulates the height of the drop of the horizontal arm, and thereby also the effect of the bumping action upon the feed. A small lever at one side of the adjusting-bolt permits the horizontal arm to be raised above the

level of the cam, and in this way the feed can be instantly stopped and started.

It has been a common experience that as rolls are usually fed, even when great care is taken to secure a feed-sheet of uniform thickness, the roll-shells will groove and wear more rapidly at their centers than at their ends. This is frequently caused by a difference in the rate of flow of the feed-stream, owing to friction of the sides of the chute or spout leading from the hopper to the rolls. Another cause of greater wear of the central portion of the roll-shells is the greater mobility of the feed-stream in certain directions than in others, when subjected to a crushing pressure. This difference in mobility causes a difference in the amount of abrasion, and it shows itself by the cupping, grooving, or corrugating of the roll-faces along the center, and by ridging at the edges of the roll-shell. Mr. Frazee has found that the difficulty of irregular wear of the roll-shells can be entirely overcome by a close control over the sectional shape and area of the feed-stream, and by feeding a greater amount of material along the sides of the roll-faces than at their centers.

As shown more clearly on a larger scale in Figs. 5 and 6, the projecting side of the angle-iron which forms the lip of the feed-tray is cut down at its two ends, so as to slope gradually towards the ends, leaving its full height only at the center. In this way the feed-stream passing over the lip or dam is made thicker at the ends than at its center. Furthermore, to regulate the flow and upper contour of the feed-stream, as well as the lower, a guide-plate, which is usually hinged to the sides of the feed-tray, is allowed to rest upon the surface of the feed-stream. This is provided with a wearing-plate at its lower edge, and is so beveled at its two ends as to permit a somewhat greater freedom of flow of the feed-stream at its edges than at its center. The shaded area in Fig. 6 shows the approximate cross-section of the feed-stream, which is adjusted to secure an equal abrading effect across the entire width of the roll-faces, and thus to maintain them in perfect parallelism while they undergo wear. The exact cross-section of the feed-stream will vary somewhat with the hardness of the different materials to be crushed. This can be easily adjusted by varying the amount of bevel given to the edges of the angle-iron

and of the guide-plate in the feed-tray until the feed-stream exerts the desired uniform abrading action.

The flanges upon the longer roll-shell perform two functions. They not only confine the feed-stream at its ends, and prevent ridges from forming at the ends of the unflanged roll-shell, but by carrying the feed-stream a slight distance beyond the end of the shorter roll-shell, a small amount of crushing is performed between the sides of the flanges and the ends of the roll-shell. This flange-crushing, being in opposite directions, balances, and all tendency to exert end-thrust of the shafts against their bearings is thus neutralized and overcome. By the simple means thus provided it becomes possible for the roll-tender to maintain the roll-faces parallel while crushing even the hardest material, and to keep the rolls in continuous operation until the roll-shells are completely worn out. The only assistance required of a machinist is occasionally to turn off the edges of the flanges when they become so long that they strike the bolts of the opposite roll which hold its roll-core in place.

Figs. 7 and 8 illustrate the effect of Mr. Frazee's invention in connection with 12-in. and 24-in. roll-shells, in keeping their surfaces cylindrical until the shells are completely worn out. The 12-in. roll-shell shown in Fig. 7 has been reduced to 6.75 in. in diameter.

The 24-in. roll-shells shown in Fig. 8 have been reduced to 19.25 in. in diameter, leaving a thickness of only $\frac{5}{8}$ in. of metal at their edges. At their centers they are somewhat thicker, owing to their beveled inner surfaces, which are required in order to mount them upon their coned centers. A pair of the 24-in. crucible-steel roll-shells, 14.5-in. face, when machined and ready for use, weigh approximately 1,776 lb. When worn down to the size shown in Fig. 8, their weight is 438 lb. The difference—viz., 1,338 lb., or 75.3 per cent. of the original weight—has been entirely expended in useful and effective work. Such a novel and valuable result, whereby the efficiency of the crushing-rolls is maintained uniform while the roll-surfaces undergo wear, clearly marks a step in advance in the art of roll-crushing. The only machine-work required, as already explained, consists in a partial removal of the flanges on the longer roll-shell as the wear of the shells per-

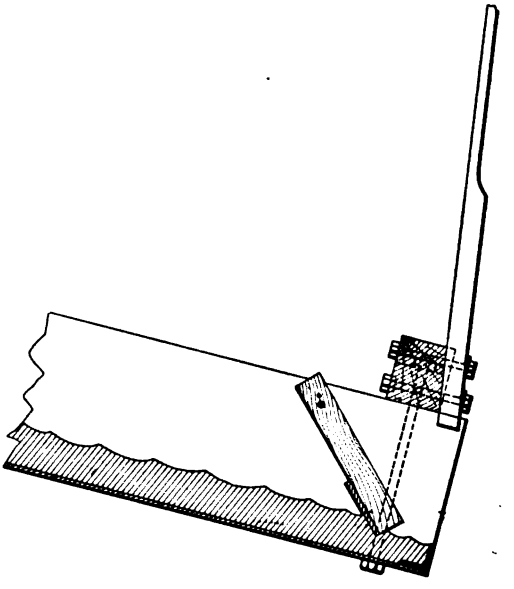


FIG. 5.—LONGITUDINAL SECTION OF END OF FEED-TRAY.

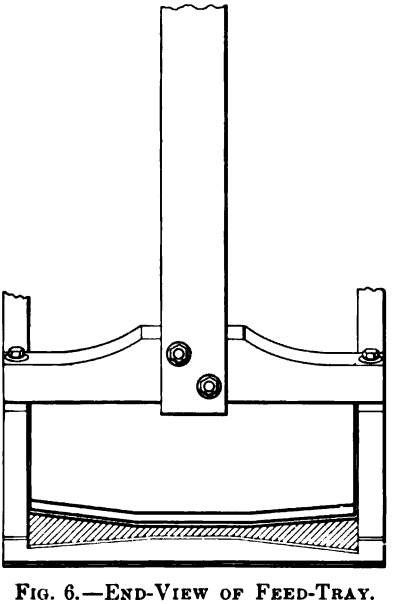


FIG. 6.—END-VIEW OF FEED-TRAY.

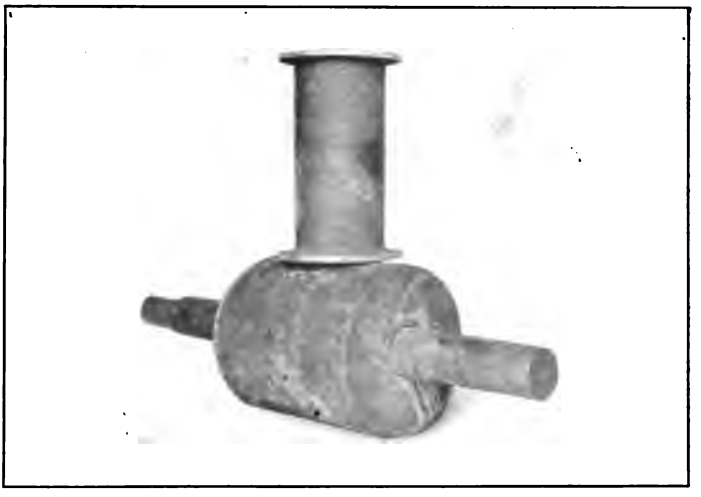


FIG. 7.—A 12- BY 14-IN. ROLL-SHELL, MOUNTED UPON ITS SHAFT, AND A FLANGED ROLL-SHELL OF THE SAME ORIGINAL SIZE WORN DOWN TO 6.75 IN. IN DIAMETER, ITS SURFACE BEING MAINTAINED CYLINDRICAL WITHOUT ANY MACHINE-WORK HAVING BEEN EXPENDED UPON IT.

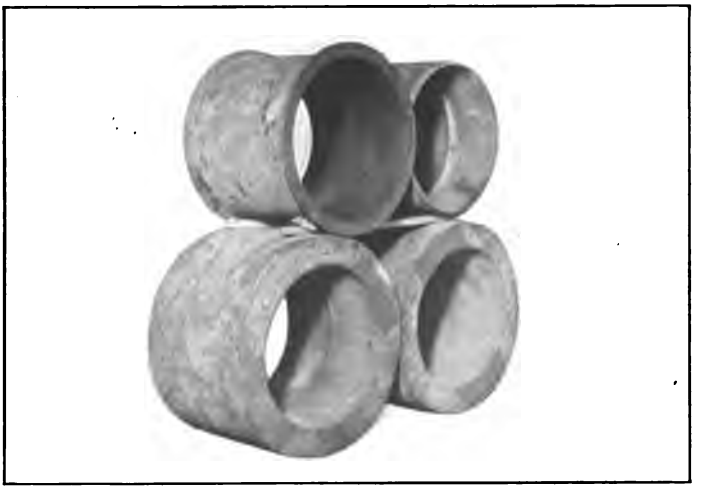


FIG. 8.—A PAIR OF WORN-OUT ROLL-SHELLS ORIGINALLY 24 IN. IN DIAMETER AND REDUCED TO 19.25 IN. IN DIAMETER BY WEAR ALONE, PLACED ABOVE A PAIR OF 24-IN. STEEL ROLL-SHELL CASTINGS.

mits the flanges to approach the draw-bolts of the coned centers.

While the method of feeding the rolls which Mr. Frazee has adopted accomplishes admirably the purpose for which it was designed, and secures a close control over the feed-stream at its point of discharge, it is evident that the same principle might also be adapted to other types of feeders, such as the roller-feeder, the shaking-feeder, the plunger-feeder, the scraping-feeder, etc.

This new roll-design and manner of operating rolls has been developed by Mr. Frazee in connection with the dry crushing of very hard materials used for abrasives, such as quartz, garnet, etc. It is equally applicable to ores, and by the settling and removal of water from the mill-feed, it is possible to apply it to wet crushing as well as dry. The field which is thus opened up in connection with ores which require careful granulation, in order to prepare them for concentration, is a very large one.

Perhaps the most interesting and obvious application of Mr. Frazee's new roll-design is in connection with the treatment of complex ores. Innumerable processes have been proposed for the recovery of the values contained in these perplexing ore-mixtures, but they may be divided broadly into two classes, viz., that which includes mechanical methods, and that which includes all the others, such as chemical processes, leaching, smelting, etc. While the former are the least expensive, the results heretofore obtained have been proximate only, and the recoveries have shown a low efficiency. All of these mechanical processes, whether they employ gravity, magnetic, electrostatic, or flotation methods (with the single exception of the flotation process adopted by the Minerals Separation Co., Ltd.), require an ore to be crushed to a point where the separate minerals it contains are sufficiently unlocked to permit them to be mechanically separated from each other, so that they may be collected in different groups. The point does not seem to have received sufficient attention, that any one of the above mechanical processes of treating complex ores can be seriously interfered with, if not completely upset, by the means employed to crush such ores in preparing them for treatment.

The mineral association of these mixed sulphide ores is usually a very intimate one, so much so that in many cases it

involves crushing them to pass a 30- to 40-mesh screen, or even finer, in order to sufficiently unlock the associated minerals.

By the ordinary methods of crushing there is great danger of producing a large amount of fine powder, or slime, in crushing to such sizes. Particles of a very minute size, usually placed at about 200-mesh, then become, in a measure, a law unto themselves, and the losses in treating such material are very great. The obvious way to solve the dust, or slime, problem is not to make any dust, or slime, or as little as possible, in crushing these ores for mechanical separation.

The future of the mechanical processes of ore-treatment, in competition with others which promise larger returns, will thus depend, in many cases, upon successful granulation. In this direction I believe that Mr. Frazee's invention, as outlined above, has accomplished a great step in advance in the art of roll-crushing, and it holds out great hopes for the future in the more successful preparation of many ores for mechanical concentration, by reason of the control he has secured over the wear of his roll-shells, whereby a better granulating action can be obtained.

Abrasion and Dust-Losses in Ore-Drying.

BY CARL F. DIETZ AND DYKE V. KEEDY, BOSTON, MASS.

(New York Meeting, February, 1912.)

THE problem of drying ores is one that most mill-engineers are sooner or later called upon to meet, and it may be timely to point out some of the difficulties resulting from such operations from purely a metallurgical and economic standpoint. We mean, in the following pages, to discuss primarily the effect of such operations on the ore itself, without particular reference to the mechanical or cost features of the subject. That an appreciable tonnage of the ore fed to the usual form of drier, either rotary or stack, is lost, is fairly well appreciated, but we believe that only a few mill-managers know just what their drier-losses amount to, and we have found that in most cases they console themselves with the fact that the drier-smoke is not very much discolored, and it is generally believed that the draft is not sufficient to carry away more than a negligible percentage of the values.

The drying of ores is becoming an important factor since the more general application of magnetic, static, and dry-table treatment. Its importance in the field of metallurgical operations warrants a close examination into its effects. This subject has been repeatedly brought home to us, and while the drier-losses, in every case examined, have considerably exceeded even the most generous estimate, an attempt to recover these values is rarely made. We have found that the losses resulting from drying-operations can be effectively recovered, at a low cost, and believe that the publication of the results of a number of investigations along these lines will be of interest and benefit to the profession at large.

To dry an ore quickly and effectively, it must be brought into actual contact with hot gases or air. The rapid drying is effected by the fact that the hotter the gas the hungrier it is for moisture. A consideration of the following data indicates the rapidly increasing moisture-carrying capacity of air or gas, at atmospheric pressure, as the temperature increases:

Temperature. Degrees F.	Weight of Water-Vapor Mixed with 1 lb. of Air. Pounds.
32,	0.003744
82,	0.023526
132,	0.118548
182,	0.728012
192,	1.253190
202,	2.855070
212,	Infinite.

The types of driers usually employed are: (1) stacks of brick or iron in which the ore falls over baffle-plates, against the ascending current of hot gases, and (2) horizontal rotary tubes, in which the ore is propelled, due to the increasing diameter of the conical shell, or to a slight pitch when the tube is a cylinder; or to either of these in addition to a series of spiral angle-plates attached to the interior of the shell, acting as a screw-conveyor.

In the rotary driers, it is usual to have lifting-angles on the inner wall of the shell for picking up the ore and carrying it to a varying height, allowing the material to fall to the bottom of the cylinder through the current of hot gases. Thus a myriad of surfaces is presented from which the moisture may be evapo-

rated. In the stack drier, the same effect is produced when the ore drops from baffle to baffle, thereby being spread out into a rather thin layer.

While under these conditions the drying-factor is a fairly efficient one, yet two other effects, not desirable, result. The first of these is abrasion, while the second is the loss due to the draft carrying along fine particles of value. The abrasion-factor is practically the same in the two types of driers unless in the rotary type the passage of the ore is very slow and each piece is permitted to fall many more times than it would in passing from baffle to baffle in the stack drier. That the abrasion in the drying of sulphide ores is a large factor, and that as a rule the mineral values abrade the more readily, is generally admitted; but few engineers realize the extent to which driers act as a gravity impact-crusher or tumbling-mill. If the feed to a rotary drier is carefully examined and compared to the discharge, the shape of the pieces of ore will be found to be quite different; the feed being sharp-edged and rough, as against the discharge showing many smooth surfaces, while at the same time the rough and sharp edges have almost wholly disappeared. Abrasion is the cause, and of course the heavier the rock the greater will be the impact, and consequently the greater the abrasion. There is, therefore, an economic limit to the size of the material to be dried, entirely aside from the factors of damage to plates or shelves, or the danger of not thoroughly drying through to the core, in the case of porous material. On the other hand, a fine feed has the decided disadvantage that the operation produces much dust, resulting in a great loss.

Such drier-losses first made themselves uncomfortably noticeable to us in the operation of a dry milling-plant in Utah handling lead-silver ore in which two-thirds of the lead was in carbonate form, while fully 10 per cent. of the silver-values occurred as halogen compounds.

The ore was crushed to about 1.5 in. in size before entering the drier. This consisted principally of a conical shell 24 ft. long and about 4.5 ft. in diameter, rotating at 8 rev. per min. The draft was just sufficient to take care of a light fire, all that was necessary, since the ore carried but from 5 to 8 per cent. of moisture. Even under such conditions and after taking out

the alternate lifting-angles, the gases issuing from the top of the stack showed, by their color, that an appreciable quantity of fine dust was being lost. The back connection of the drier consisted of a large chamber, the flue from which to the stack was very liberal in cross-section. Hence the flow of the gases was so very slow that most of the dust was permitted to settle in the chamber and in the bottom of the smoke-stack. A fairly close approximation of the distribution of the ore after it was dried is found in the following:

Products.	Weight. Per Cent.	Assay. Pb.	Assay. Ag.	Percentage of Total. Pb.	Percentage of Total. Ag.
Total feed, ...	100	10.35	9.68	100	100
Dry discharge, ...	96.2	9.77	9.34	90.7	95.40
Settled in chamber, . . .	1.77	35.30	10.30	6.04	1.94
Settled in smoke-stack, . .	0.76	18.50	8.60	1.36	0.69
Lost in gases, ...	1.27	14.30	15.20	1.75	2.04

From the above it will be seen that of the total feed to the drier, 3.8 per cent. by weight was withdrawn by the draft, corresponding to a loss of 9.3 per cent. of the lead-content and 4.6 per cent. of the silver-content. At the then prevailing metal prices, 7.6 per cent. of the total value in the ore was taken out by the draft. Of this, the greater portion, however, due to the generous dimensions of the settling-chamber, flue, and smoke-stack, was recovered in the form of a somewhat finely-divided product of sufficiently high grade to be mixed with the concentrates, resulting in a final loss of unrecovered material, going off as an impalpable powder in the stack-gases, of 1.75 per cent. of the total lead and 2.04 per cent. of the total silver, or a money-value of 1.85 per cent. of the total values in the ore.

Had it not been for the precautions employed in operating the drier and crushing no finer than 1.5 in., the final losses would have been much heavier and would have necessitated treating the drier-gases for a recovery of the dust. The slow movement accounts for the reduction in lead-tenor as the products grow finer, while the silver-values manifest their finely-divided form by their presence to the greatest value in the dust of the discharge-gases.

In the case of a cyanide-plant in southern Utah, crushing gold-ores dry, prior to cyanidation, the ore came from the mine

in a very wet state. Consequently four rotary driers were driven hard to effect drying the tonnage necessary. The ore was crushed to but 2 in. in size before going to the drier-plant. The smoke issuing from the stack was light gray in color, due to the quartz, and on quiet days settled its burden over the immediately surrounding country. From 5 to 6 per cent. of the tonnage was lost regularly in this operation; and assays of the dust contained in the gases as issuing into the atmosphere ran 0.48 oz. of gold and 0.35 oz. of silver, derived from an ore carrying 0.31 oz. of gold and 0.15 oz. of silver. This dust was all finer than 150-mesh, but could readily have been recovered and treated. The loss suffered by this drying-method amounted to approximately 8 per cent. of the values in the ore.

Two other notable cases may be mentioned in passing; one, in northern Norway, where mixed magnetite and hematite ore carrying 35 per cent. of iron was being milled on a very large scale. The rock, after having been crushed to about 3-in. size, passed through a set of stack driers, which at certain times in the year, when snow and ice came down with the ore from the quarries, it was necessary to operate with forced draft. While the dust-loss was not actually measured, it is safe to say that it was not less than from 7 to 10 per cent. by weight. In addition to this, the assay-value of the dust was considerably higher than that of the crude ore, since the specular hematite, being friable, suffered the greatest abrasion. The surrounding country for hundreds of feet from the drier-house was covered with fine iron-ore, and the smoke from the drier-stacks was a dark brown and just as dense as that issuing from a chimney directly after boilers have been fired with soft coal. The trouble in this case was a lack of drying-equipment; and since the time that ore is exposed to the action of the gases in a stack drier is not readily controllable, a sharp increase in moisture-content of the ore necessitates driving the driers with forced draft, resulting in a very much heavier dust-loss than would be suffered if the upward-moving gas-current had a lower velocity.

Another instance in one of the far western States, where sulphide-losses were heavy, may be mentioned. The material treated was a middlings product consisting of a zinc-blende and iron pyrite, practically all of it finer than 16-mesh, and

carrying from 20 to 30 per cent. of moisture. The separating process used, being a dry method, required the drying of the feed so that it would lend itself freely to fine sizing. The type of drier used was a rotary cylinder through which the gases passed in contact with the ore, and fearing a considerable drier-loss, a settling-chamber was installed between the back connection of the drier and the smoke-stack. The deposition of fine sulphide-dust in the settling-chamber was relatively very large, but frequently this was not deposited in the dry state, since a reduction in temperature of the gases in this chamber permitted some of the moisture to condense, and left a large portion of the settled product in the form of a thick sludge. Much of the fine dust escaped with the gases, as was evidenced by the color of the smoke and the rapidity with which particles settled out therefrom. The losses sustained must run into considerable tonnages in the course of a year's operation, and while we have no figures to present with regard to this particular plant, we feel safe in saying that by analogy, and our experience in other operations of a like nature, the loss cannot be less than from 3 to 5 per cent. of the total value of the ore.

In such operations where the drying of ores becomes a necessary step, precautionary measures should be taken to recover the values that would otherwise be lost in this operation.

In a more recent case where a complex zinc-iron ore (sulphides and oxides mixed) was to be treated by a dry method, it became necessary to dry the material after it had been crushed to about 1.5-in. size, driving off a relatively small percentage of moisture contained therein. This moisture would vary from 2 to as high as 10 per cent., although more generally it was in the neighborhood of from 3 to 4 per cent.

The type of drier used was a rotary cylinder 30 ft. long, 4 ft. in diameter, set at a pitch of 0.25 in. to the foot. The inside wall of the shell was equipped with numerous lifting-angles which effected a frequent dropping of the ore through the current of hot gases, but which with a relatively dry material had the disadvantage of causing more abrasion and stirring up more dust than was necessary, with a consequent loss that far exceeded all expectations. That losses would be suffered, due to the drying-operation, especially since the draft was induced by a small exhaust-fan, was to be expected, and it was deter-

mined that before any dust-collecting installations were to be made, the actual loss sustained in this operation was first to be measured. As a precaution against an excess loss of the dust, the draft-fan was operated with just enough speed to carry away the gases from a very light fire. The fan was so accurately adjusted that a reduction of but 15 to 20 rev. per min. caused some of the furnace-gas to leak through the brick setting and other openings, into the mill.

We had here an ore that in its general make-up is representative of a very large class of material that engineers are frequently called upon to dry. The ore carried about 20 per cent. of zinc as its principal value. This zinc was mostly in the sulphide state, having some iron chemically combined with it, so that the individual crystals varied from a light semi-translucent brown to quite a dark brown, opaque in appearance. The iron associated with the zinc was present primarily in three forms; that is, the sulphide, specular hematite, and magnetite. The percentage of iron in the ore was about equal to that of the zinc. As to other values, there was a small quantity of lead as galena carrying a few ounces of silver to the ton. The gangue consisted primarily of silica and some lime. Some of the mineral values were oxidized, considerably further complicating the mixture, and there were present in appreciable quantities some carbonates of zinc, lead, iron, and manganese.

Extent and Effect of Abrasion.

The type of drier in use at this plant was faulty in at least one detail, that being the rather unfortunate position of the feed with respect to the gas-outlet. The feed-pipe discharged its ore into the drier-shell at just about the point where the velocity of the gases was greatest; hence the power to draw fine material into the stack even before it came in contact with the drier-shell was great. Even with ore carrying from 10 to 15 per cent. of moisture, a heavy loss was experienced, and with actually wet ore the conditions were not much improved, since it thus became necessary to drive harder to effect the same degree of dryness. Hence the velocity of the gases passing through the shell was sufficient to carry along particles of considerable size and the net results were much the same.

Since superficial investigations indicated a very heavy abra-

sion-factor in the drier, which in part resulted in heavy tonnage- and zinc-loss, it became desirable to have a reasonably accurate measure of this factor. A test to determine this was carried out as follows:

It was attempted to have the conditions during the test as nearly as possible identical with those existing during regular mill-operations, the only difference being that the escape of dust should be minimized to the greatest possible extent, so that as nearly as possible all the material fed to the drier would be accounted for in the products derived therefrom. Even though the ore was practically dry, the drier was thoroughly heated before the test began. During the time of the test, fire-doors and ash-pit doors were closed up tightly, and the fans, both force and draft, were stopped, so that the only draft passing through the machine was such as would be induced by the tendency for the hot air within the drier to rise through the gas-connections and fan-casing, and cause fresh cold air to be sucked in through the leaks and other openings.

Precaution was taken at the discharge end of the drier so that contamination between the ore on which the test was being made and the material piled up in the back connection could not take place.

A measure of the abrasion can be had only by comparing the screen-analysis of the material when fed to the drier with the screen-analysis of the same material as it issues therefrom. For the purpose of producing as definite results as possible, the crude ore to be fed to the drier consisted of pieces no larger than those passing through the Blake crusher, about 2-in. ring, while also everything finer than 16-mesh was eliminated. Thus a direct relationship is to be had, since any material finer than 16-mesh in the discharge from the drier must necessarily have been produced during the passage of the ore through the machine. The ore as it was discharged from the drier was again screened over 0.25-in. and 16-mesh screens, the same as the feed. The relationship between these screen-analyses directly measures the abrasion-factor.

During the time of the test the stack-draft, through the drier, carried out into the atmosphere a small proportion of very fine material, all of which was —200-mesh, and which represented the actual quantity not accounted for between the feed and the discharge. On account of the fineness of this

dust, this must be charged to the —200-mesh material in the screen-analysis of the dried ore. The results of this test are as follows:

The test was made on 2,177 lb. less 0.8 per cent. of moisture, or 2,160 lb. of dry ore, of which

83.7 per cent. was — 2-in. ring + 0.25-in. mesh.
16.3 per cent. was — 0.25-in. mesh + 16-mesh.
0 per cent. was — 16-mesh.

The ore discharged from the drier was 2,110 lb., with negligible moisture-content, showing a loss of 50 lb. of ore during the test, or 2.3 per cent., which must be charged against the —200-mesh product. This loss was represented by the fine material carried away by the draft. The dry ore discharged showed as follows:

Dry Ore.	Pounds.	Per Cent.	Interpolated when Adding Loss to —16-Mesh Product. Per Cent.
+ 0.25 in., . . .	1,611.0	76.3	74.4
— 0.25 in. + 16-mesh, .	288.5	13.7	13.3
— 16-mesh, . . .	210.5	10.0	12.3
	2,110.0	100.0	100.0

From the above data it will be noted that 12.3 per cent. of the total ore was ground into sizes finer than 16-mesh simply by the tumbling action taking place in the drier.

A screen-analysis of the —16-mesh material in the drier product and interpolated with the loss charged to —200-mesh float, as well as this screen-analysis applied to the total dry ore, shows:

Screen-Analysis on —16-Mesh Material of Dried Ore with Loss of 2.3 Per Cent. Added to Float. Per Cent.			Screen-Analysis of Total Ore After Passing Drier. Per Cent.
		+ 0.25 in.	74.400
	— 0.25 in.	+ 16 mesh.	13.300
11.74	— 16	+ 20	1.420
10.17	— 20	+ 24	1.240
8.00	— 24	+ 28	0.970
5.34	— 28	+ 34	0.650
8.00	— 34	+ 42	0.970
6.44	— 42	+ 50	0.790
6.53	— 50	+ 62	0.797
5.57	— 62	+ 74	0.695
4.78	— 74	+ 86	0.585
9.30	— 86	+ 109	1.146
3.40	— 109	+ 125	0.422
4.40	— 125	+ 150	0.538
7.03	— 150	+ 200	0.862
6.86	— 200	grit	0.838
2.44	— 200	float	0.293

NOTE:—The —200-mesh material was treated in an up-cast, dividing the product into grit and float, the former representing a good table-feed, while the latter consisted of real slime, difficult, if not impossible, to treat mechanically.

From the above screen-analysis it will be seen that the drier acts as a tumbling-mill, reducing a considerable proportion of the ore, relieving the crushing-department of 12.3 per cent. of the work it would otherwise be called upon to perform. On the face of things, in the event of the abraded material being recovered and not lost with the drier-gases, this action might appear to be advantageous. The abrading action of the drier, however, throws at least 0.3 per cent. of the total ore into a product which becomes practically useless from a separating point of view. Since in the test all the original fines had been eliminated from the feed, it is likely that the quantity of —200 float in the drier-discharge is more nearly 1 per cent. on the total ore. By weight this is not so serious, but the assay of these products shows that the heaviest abrading effect takes place on the zinc. It was found that with the crude ore assaying 19.1 per cent. of zinc, the abraded or —16-mesh portion of the dried product as discharged from the drier showed an assay-value of 29.4 per cent. of zinc. The —200-mesh material contained in the dry ore shows a zinc-value of 26.1 per cent. We have then as follows:

Pounds. Per Cent.

2,160 @ 19.1 = 412.5 lb. of zinc in ore.
2,160 @ 12 3 = 266 lb. of ore-abraded material.
266 @ 29 = 78.2 lb. of zinc in abraded material.

$$\frac{78.2}{412.5} = 18.9 \text{ per cent. of zinc abrasion.}$$

From this, the abrading effect, while the ore is passing through the drier, is 50 per cent. greater on the zinc-mineral than it is on the total ore, proclaiming the zinc as the greatest sufferer.

In the event of the dust thus made being lost, the abrading effect of the drier becomes very serious, while if the dust is recovered and disposed of in the proper places in the system, this effect, while not to be particularly desired, does not represent so disadvantageous a factor.

The effect of the drier-abrasion will be pointed out more in detail in recording a test on the entire reduction and dust-collecting portion of the plant in question, but at this point it may be well to point out three ways in which the drier-abrasion may be very materially reduced.

(1) Reducing the number of lifting-angles in the shell, which will cut down the number of times the ore is dropped.

(2) Increasing the pitch of the shell to 0.5 in. per foot, from 0.25 in. This will cause the ore to travel through the drier at practically double the speed.

(3) Introducing both No. 1 and No. 2 methods simultaneously.

The increased pitch of the drier-shell would result in a larger capacity, however, at the expense of the efficiency of the machine when moisture in the ore runs high.

Recovery of Dust-Losses.

This particular plant, in which an entirely dry process was being operated, was equipped with an extensive dust-system operated by two 45-in. steel-plate exhausters, which, by suitable connections and draft-gates, caused a slight vacuum in the housing of all crushing-machines, as well as at all points where ore was handled, such as in the boots and heads of elevators, chutes, bins, and feeders. In this manner, with a properly adjusted suction-system, any dry-plant can be kept effectively clear of the obnoxious dust, with the result that instead of causing injury to attendants (by either the danger of lead-poisoning or phthisis), or damage to shafting, bearings, pulleys, and belting, this product, which is in most instances a valuable one, is collected and subject to treatment; thus, instead of suffering the losses as pointed out, at least a large portion of them may be recovered. The dust collected throughout the mill was discharged into a centrifugal dust-collector, Fig. 1, which was at first operated dry, but its efficiency was not as high as was desired. The velocity of the air was somewhat greater than that which would have permitted practically all of the dust to settle out, so that the apparatus under such conditions worked as a very excellent separator, inasmuch as the collected product was of a gritty texture, even though practically all of it would pass through a 200-mesh screen. Only the very finest of the dust passed out through the stack of the collector, and this was almost pigment-like in physical characteristics, resulting in no gritty feeling whatever, when rubbed between the fingers.

This type of dust-collector we have used with considerable success in several cases where very much larger volumes of gas were being handled, and found that to improve the efficiency the introduction of very fine water-sprays or steam-jets was most effective. The steam when condensing, or a very finely divided water-spray, acts as a powerful scrubbing-agent, wetting the particles of dust, thus enabling them to be caught by the film of water on the sides of the collector when the centrifugal force causes them to impinge against it.

To improve the efficiency of this dust-collector, both steam and water were introduced, using for this purpose a portion of the exhaust steam from the main engine. This was delivered into the inflow-pipe and in the direction of the flow of the air, while three jets of water divided into very fine sprays by means of an ordinary garden-hose nozzle were fitted into the sides of the collector about on a level with the center of the inflow-pipe. The bottom of the collector was sealed by about 6 in. of water, and all of the product, consisting of a thin sludge, was collected in a settling-tank below, from which the excess water was allowed to run to waste. The effect of this system was astounding, inasmuch as the collection of the dust was practically perfect, the air issuing from the top of the collector being barely discolored by the presence of any solid particles, and all efforts at collecting such escaping dust were unsuccessful, even by filtering through cotton. The tank collecting the slimes acted as a thickener, allowing the excess water to overflow, from which a slime-table feed could be drawn, while the excess water could either be returned to the spray-nozzles or be permitted to flow to waste.

For the purpose of collecting the drier-dust losses, the draft-fan of the drier was by-passed and the gas-outlet equipped with a blast-gate and connected with a large chamber, from which the exhausters received their dust-laden air. All of the dust-pipe connections throughout the plant lead to this central chamber, which for lack of a better term may be called "the vacuum-box," and which, due to its volume and the consequent reduction in velocity of the air or gas passing through it, permitted some of the heavier dust to settle.

Test of Dust-Collecting System.

With provision thus made for collecting not only the relatively small quantity of mill-dust but also the large amount of ore lost in the drier-operations, a complete test of the drying- and crushing-plant with dust-collecting system in operation was carried out. This test gave some rather remarkable and interesting data, and fully demonstrated, not only the extent to which milling-operations may suffer, due to a loss of the drier- and mill-dust, but also the value of a simple and efficient dust-collecting system. The draft was again regulated so as just to prevent the escape of any furnace-smoke into the mill.

Since any appreciable escape of valuable dust was no longer possible, such material as was not accounted for by the feed to the table-department could only distribute itself in three different products: *i. e.*, vacuum-box, dust-pipes, and slime-box. These three products were carefully collected and weighed after the test, everything having been thoroughly cleaned before the test began. These, as also the concentrator-feed, were carefully sampled, screen-analyzed, and assayed, so that a full and complete distribution of the zinc in the various products could be arrived at, as well as an ultimate screen-analysis of the total ore.

In the screen-analyses there is recorded in each case a —200 grit and a —200 float-product. These figures were obtained in each case by agitating the —200-mesh material in a beaker, while at the same time a small stream of water was allowed to flow into the beaker, thus overflowing, and carrying with it the finer particles of the —200-mesh product. In this manner this product was divided into what might properly be termed "grit" and "slime." However, the action was more intense than it would be in a settling-apparatus, and hence shows a slightly higher proportion of float material than actually exists in the crushed ore.

Seven thousand pound of crude ore were weighed out and fed through the Blake crusher to the drier. The crude ore contained 0.7 per cent. of moisture, accounting for about 50 lb., which, however, is not taken into account in the following figures, since the factor of error, in weights, collection of products, and observation, is probably at least as large as this moisture-factor.

These 7,000 lb. of ore were distributed in the four products as follows:

	Pounds.	Per Cent.
Concentrator-feed, ...	5,861	83.6
Vacuum-box, ...	601	8.6
Suction-pipes, ...	24	0.34
Slime-box, ...	514	7.34

Table I. shows the distribution of the ore, by weight, in the various sizes in each of the four products, and the assay-value of zinc in each of the products resulting from the screen-analysis.

TABLE I.—*Percentage-Weight and Value of Ore in Various Sizes of Each Product.*

	Concentrator-Feed.		Vacuum-Box.		Suction-Pipes.		Slime-Box.	
	Weight	Zn.	Weight.	Zn.	Weight.	Zn.	Weight.	Zn.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
— 16 + 20....	19.—	14.1						
— 20 + 24....	11.—	17.3						
— 24 + 28....	9.50	17.9						
— 28 + 34....	6.25	19.6						
— 34 + 42....	8.25	19.2	0.50	10.5				
— 42 + 50....	7.—	19.1	1.75	15.3				
— 50 + 62....	6.—	20.6	3.75	19.—				
— 62 + 74....	4.75	21.8	7.25	20.8				
— 74 + 86....	3.—	21.3	10.75	22.4				
— 86 + 109....	6.50	20.6	25.—	26.6				
—109 + 125....	2.50	20.1	8.—	27.—			7.—	18.5
—125 + 150....	2.50	20.4	8.—	25.7	5.—	20.2	5.—	21.5
—150 + 200....	3.50	21.8	14.50	23.8	8.—	20.5	12.5	22.8
—200 Grit.....	6.35	20.9	11.27	27.4	57.85	19.35	11.1	21.-
—200 Float.....	4.65	17.1	9.23	21.3	29.15	18.50	64.4	19.6

In order that a complete screen-analysis of the combined products, as well as a distribution of the actual zinc-content of the ore in the various products, may be obtained, Table II. was prepared, showing actual distribution by weight of the material and the zinc-content, from which is calculated the average zinc-analysis of each of the four products made.

TABLE II.—*Actual Weight and Values of Ore in Various Sizes of Each Product.*

	Concentrator-Feed.		Vacuum-Box.		Suction-Pipes.		Slime-Box.		Total.	
	Pounds.	Zn.	Pounds.	Zn.	Pounds.	Zn.	Pounds.	Zn.	Pounds Ore.	Pounds Zn.
+ 20.....	1,112	156.0							1,112.0	156.00
+ 24.....	645	111.5							645.0	111.50
+ 28.....	551	98.6							551.0	98.60
+ 34.....	363	71.0							363.0	71.00
+ 42.....	480	92.0	3.0	0.35					483.0	92.35
+ 50.....	410	78.3	10.5	1.61					420.5	79.91
+ 60.....	351	72.2	22.5	4.27					373.5	76.47
+ 74.....	270	58.8	43.5	9.05					313.5	67.85
+ 86.....	177	37.5	64.6	14.47					241.6	51.97
+ 109.....	375	77.2	150.4	39.90					525.4	117.10
+ 125.....	141	28.3	48.0	12.95			36.00	6.65	225.0	47.90
+ 150.....	141	28.7	48.0	12.34	1.20	0.252	25.70	5.52	215.9	46.81
+ 200.....	199	43.3	87.1	20.72	1.92	0.403	64.20	14.63	352.22	79.05
— 200 Gr.	372	77.8	67.8	18.54	13.89	2.685	57.00	11.95	510.69	110.97
— 200 Fl.	273	46.7	55.4	11.81	6.96	1.275	331.00	64.85	666.36	124.63
									6,998.67	1,332.11

The average of the heads is 19.01 per cent. of zinc by calculation. From the above data we then have the zinc-distribution in the various products, as follows:

	Pounds.	Per Cent.		Zinc Distribution. Pounds Zinc.	Per Cent.
Concentrator-feed, ...	5,861	@ 18.4	=	1,077.5	81.00
Vacuum-box, ...	601	@ 24.3	=	146	10.93
Suction-pipes, ...	24	@ 19.23	=	4.62	0.35
Slime-box, ...	514	@ 20.15	=	102.9	7.72
				1,331.02	100.00
Crude ore, ...	7,000	@ 19.01	=	1,330.7	

The reasonable correctness of these data is proved by the assay-value of each of these products on the original samples, from which we have to note the following comparison:

Product.	Assay from Sample. Per Cent. of Zinc.	Assay by Calculation. Per Cent. of Zinc.
Concentrator-feed, ...	18.7	18.40
Vacuum-box, ...	24.7	24.30
Suction-pipes, ...	19.4	19.23
Slime-box, ...	20.3	20.15

From the above data it is possible to set up a combined screen-analysis as well as the zinc-distribution, and also the calculated zinc-assay of each of the products in the combined screen-analysis, as shown in Table III.

TABLE III.—*Combined Screen-Analysis and Zinc-Distribution.*

Size.	Quantity by Weight in Products.	Quantity of Total Zinc in Each Product.	Zinc-Assay of Each Product.
	Per Cent.	Per Cent.	Per Cent.
— 16 + 20	15.90	11.72	14.10
— 20 + 24	9.22	8.38	17.30
— 24 + 28	7.86	7.40	17.90
— 28 + 34	5.18	5.33	19.60
— 34 + 42	6.90	6.93	19.15
— 42 + 50	6.01	6.00	19.03
— 50 + 60	5.33	5.73	20.46
— 60 + 74	4.48	5.10	21.67
— 74 + 86	3.44	3.90	21.60
— 86 + 109	7.51	8.79	22.30
— 109 + 125	3.21	3.60	21.32
— 125 + 150	3.08	3.52	21.70
— 150 + 200	5.03	5.93	22.47
— 200 Grit	7.28	8.33	21.72
— 200 Float	9.52	9.36	18.71

A comparison of the actual weight of zinc in each of the screen-products of the calculated combined screen-analysis of all of the products, and the actual weight of zinc contained in the screen-products of the vacuum-box, suction-pipe, and slime-box material taken together, gives the percentage of total zinc which did not directly reach the concentrating-department, but which was withdrawn during the drying- and crushing-operations, and collected separately in the three products mentioned.

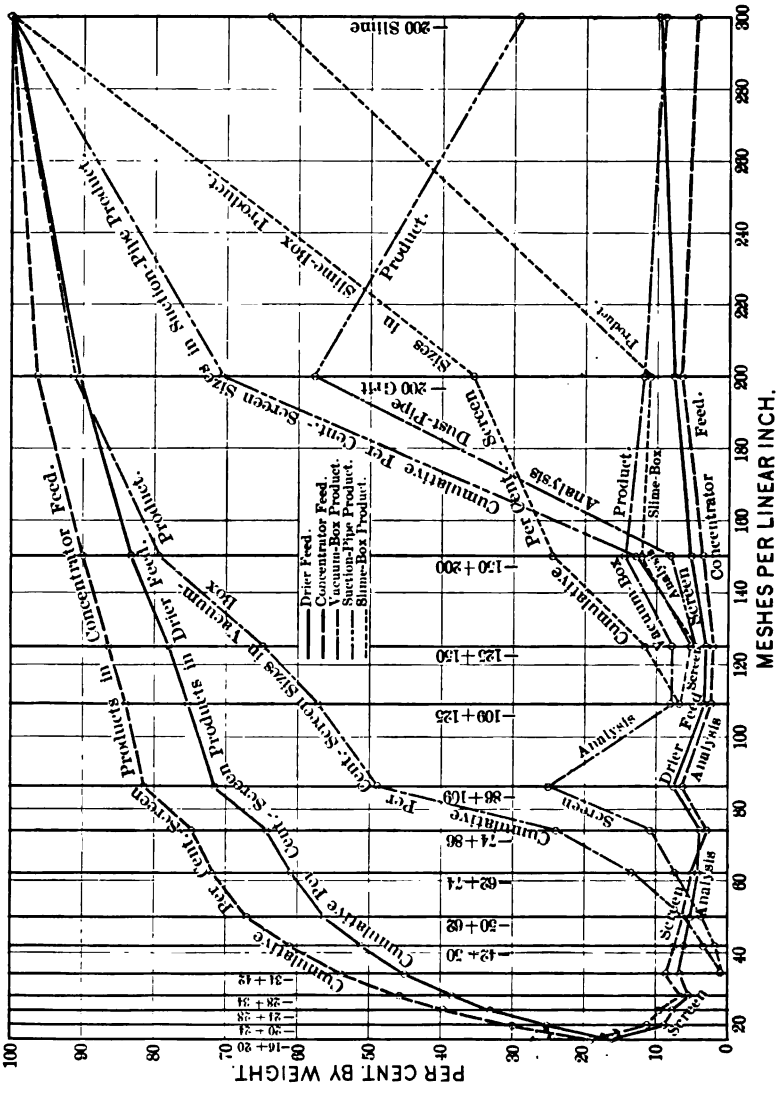
Concentrator-Feed.	Pounds Zinc.	Loss. Pounds Zinc.	Per Cent.
— 16 + 20	156		
— 20 + 24	111.5		
— 24 + 28	98.6		
— 28 + 34	71		
— 34 + 42	92.35	0.35	0.38
— 42 + 50	79.91	1.61	2.02
— 50 + 62	76.47	4.27	5.58
— 62 + 74	67.85	9.05	13.33
— 74 + 86	51.97	14.47	27.88
— 86 + 109	117.10	39.90	34.10
—109 + 125	47.90	19.60	40.80
—125 + 150	46.81	18.11	38.70
—150 + 200	79.05	35.75	45.20
—200	235.60	111.10	47.20

Of the total ore fed to the drier, but 83.0 per cent. by weight directly reached the concentrating-department, containing but 81 per cent. of the total zinc in the ore. On the whole, there

is no material effect in the direction of concentration by the drying and crushing action, but individually the three separately-collected products show marked tendencies in this respect. Particularly is this true of the vacuum-box product, in which a relatively larger proportion of the total zinc was recovered than of the ore, showing the zinc somewhat concentrated in this material. The suction-pipe and slime-box products do not greatly vary from the assay-value of the original ore.

Drier-dust losses, as well as such products as are withdrawn from the mill by means of the suction-system, are usually charged against slimes; and in the event of a complex ore are frequently considered as having been fortunately disposed of, since further treatment is difficult, if indeed, in some cases, not quite impossible. That such losses cannot be charged against the slimes, is demonstrated by the screen-analyses of the various products, which are graphically represented in Fig. 2. The legend identifies the curves, in addition to which each curve is captioned. The screen-analysis of each product is shown, as well as a curve showing the screen-analysis in cumulative percentage. Certain conditions as revealed by these curves deserve special attention. The difference in the screen-analysis between the crude ore fed to the drier and that going to the concentrating-department, as shown by the length of the ordinates between these two curves, is to be especially noted. These two curves cross at a point represented by a screen of about 65-mesh, the coarser products being in larger proportion in the screen-analysis of the concentrator-feed, while the products finer than 65-mesh are heavier in the crude ore, due, of course, to the suction-action caused primarily by the drier-draft, assisted to a small extent by the general dust-suction system throughout the plant. A distinct gradation, by size, is noted through the various products to the slime-box material, the latter showing by far the largest proportion of —200-mesh material. Each of these screen-analyses is also represented by a cumulative percentage-curve, which shows the relative fineness of the product it represents by the rapidity with which it rises as the sizes become finer. Thus the curve representing the cumulative percentage of the screen-products in the concentrator-feed is the most nearly horizontal, while the

FIG. 1.—DUST-COLLECTOR IN DRY OPERATION. NOTE THE SLIGHT DISCOLORATION OF THE OUTLET AIR AT TOP OF STACK.


PER CENT BY WEIGHT.
MESHES PER LINEAR INCH.
Drier Feed.
Concentrator Feed.
Vacuum-Box Product.
Suction-Pipe Product.
Slime-Box Product.
Cumulative Per Cent. Screen Sizes in Suction-Pipe Product
Cumulative Per Cent. Screen Sizes in Vacuum-Box Product
Cumulative Per Cent. Screen Sizes in Slime-Box Product
Cumulative Per Cent. Screen Sizes in Dust-Pipe Product
Cumulative Per Cent. Screen Sizes in Concentrator Feed.
Cumulative Per Cent. Screen Sizes in Drier Feed.
Cumulative Per Cent. Screen Sizes in Vacuum-Box Product.
Cumulative Per Cent. Screen Sizes in Suction-Pipe Product.
Cumulative Per Cent. Screen Sizes in Slime-Box Product.
Cumulative Per Cent. Screen Sizes in Dust-Pipe Product.
Cumulative Per Cent. Screen Sizes in Concentrator Feed.
Cumulative Per Cent. Screen Sizes in Drier Feed.
Screen Analysis
Screen Analysis
Screen Analysis

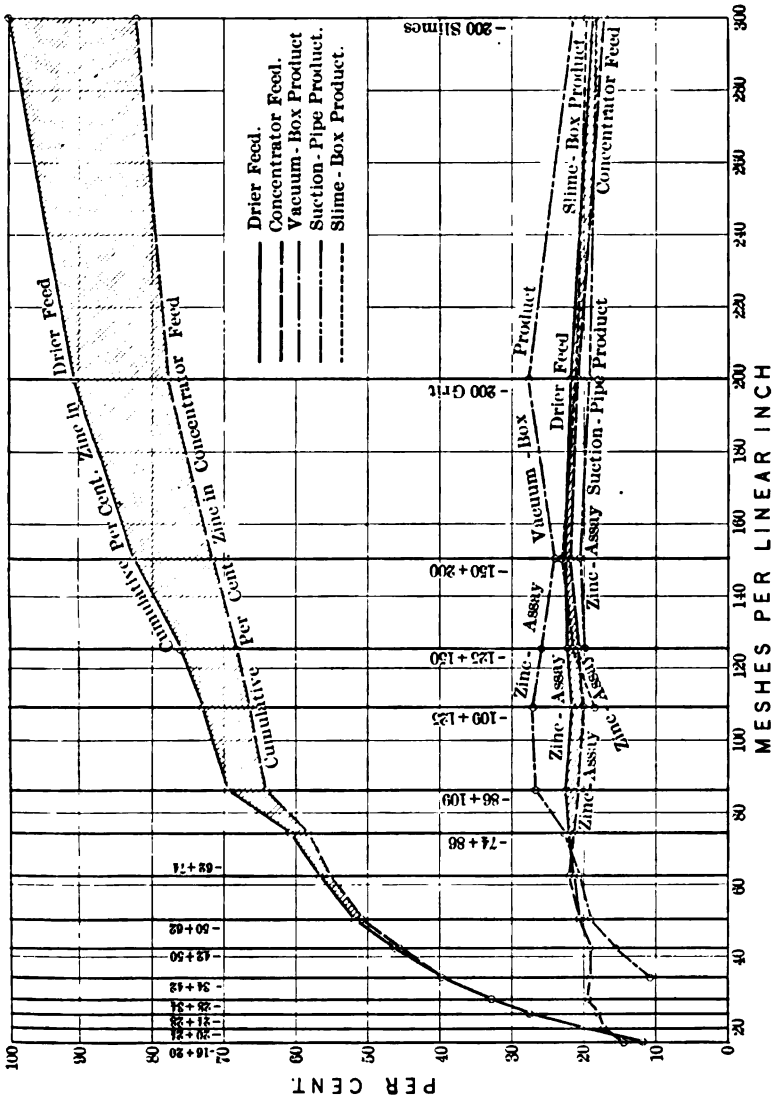


FIG. 3.—CURVES OF ZINC-ASSAYS OF SCREEN SIZES IN THE DRIER-FEED AND THE FOUR RESULTANT PRODUCTS.

curve representing the cumulative percentage of the screen-analysis of the slime-box product is the most nearly vertical.

The most important collected product to be dealt with is that which settled in the vacuum-box, and which by its screen-analysis shows but 20.5 per cent. finer than 200-mesh, of which about one-half is still of a gritty character, and represents a good concentrator-feed. That the minimum draft of this particular drier is sufficient to cause very serious losses is evidenced by the fact that the vacuum-box product contains particles as coarse as 34-mesh, and a very heavy percentage of this total product lies between 80- and 100-mesh screen. This material is in excellent form for further treatment, and, if permitted to escape into the atmosphere with the drier-

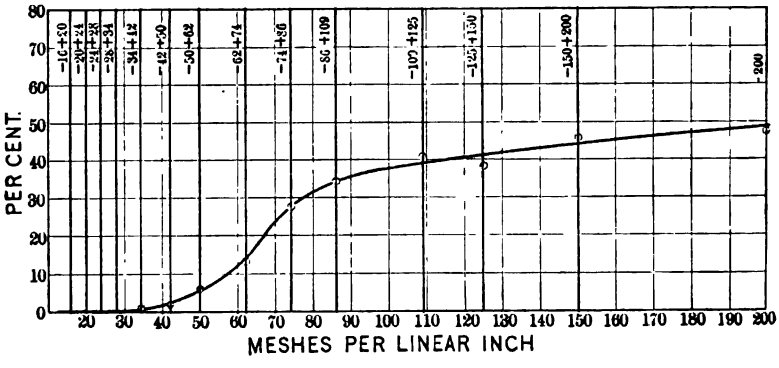


FIG. 4.—CURVE SHOWING PER CENT. OF ZINC-CONTENT WITHDRAWN DURING THE DRYING- AND CRUSHING-OPERATIONS, BASED ON THE TOTAL ORE.

smoke, entirely aside from the fact that it carries concentrated zinc-values, would spell a loss which no milling-operation can suffer without very seriously impairing its commercial possibility.

In Fig. 3 are shown graphically the zinc-assay values of the screen-sizes of all five products under consideration, as well as two curves giving the cumulative percentage of the zinc in the various screen-sizes of the drier-feed and the ore which directly reaches the concentrating-department. The sectioned area between these two cumulative percentage-curves directly represents the percentage of total zinc lost as the grades grow finer, resulting in a total loss of 19 per cent. of the zinc by the dry-

ing- and crushing-operations. The latter, however, are only a very small factor in the total.

The narrow sectioned area between the curves representing the zinc-assays of the screen-sizes in the drier-feed, as against that going to the concentrating-department, shows the loss in assay-value, due to the concentrating effect caused by the draft, which in turn is represented by the higher zinc-assay value of the screen-sizes from the vacuum-box product. The slime-box product shows much the same value as the crude ore, while the zinc-content of the suction-pipe product is somewhat lower.

In Fig. 4 is shown a curve which represents the percentage of the zinc in each screen-product of the total ore withdrawn during the drying- and crushing-operations. It is to be noted that this action does not begin on particles coarser than a 34-mesh screen, but from this point to 86-mesh the loss increases rapidly, after which the increase in loss is more gradual, still rising, however, until on the —200-mesh material a total loss of 47.2 per cent. of the zinc is suffered.

The fact that such a relatively large percentage of the material withdrawn from the drier by the draft is not in slime condition, is very satisfactory, since it permits of collecting this dust in a simple manner, to be returned to the system just before reaching the concentrating-department. Thus a material percentage of the total ore by-passes the crushing-department.

The economical collection of these fugitive values is already demonstrated and can be made a part of every dry-milling operation at very little installation-cost, while the operating-cost is so low as to be almost negligible in comparison with the benefits derived.

The Sintering of Fine Iron-Bearing Materials by the Dwight & Lloyd Process.

BY B. G. KLUGH, BIRDSBORO, PA.

(New York Meeting, February, 1912).

In a paper before the Institute at Wilkes-Barre, Pa., June, 1911,[1] Mr. James Gayley discussed the application of this process to iron-bearing materials. The same author[2] described the results of operations at the plant at Birdsboro, Pa.

The purpose of the present paper is to present further discussion of some technical details involved in the operation of this plant, and of the theoretical and practical relations of this product to blast-furnace practice.

Since Oct. 1, 1911, the Birdsboro plant has been in operation at the blast-furnace of the E. & G. Brooke Iron Co. The results have been highly satisfactory, although the material treated, being chiefly the flue-dust currently made by the furnace or taken from the accumulated stock-pile, was excessively high, and yet not uniform, in carbon, so that the desirable control or regularity of the amount of fuel in the mixture treated could not be maintained. Nothing could more forcibly demonstrate the flexibility of the process than the fact that, under these conditions, the sinter produced was, in all cases, of excellent quality.

The Dwight & Lloyd process, as such, and its present highly-developed mechanical appliances were described by Mr. Gayley in the paper above mentioned. While the repetition of some particulars will be unavoidable in this paper, it will be confined to features bearing upon the metallurgical principles involved.

According to the principle of the process, the finely-divided iron-bearing material to be sintered is intimately mixed with

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[1] *Trans.*, xlii., 180 (1911).

[2] *Iron Age*, vol. lxxxix., No. 1, p. 73 (Jan. 4, 1912).

the amount of carbon requisite to produce the sinter. This mixture is then moistened and deposited upon the machine in a uniform layer, the upper surface of which is ignited by a small flame of intense local heat, and the combustion of the intermixed carbon is effected in a progressively downward course by a current of air flowing in the same direction through the permeable mass. The energy from the combustion of each particle of carbon is expended and utilized directly upon the adjacent particles of iron-bearing material, agglomerating them into a coherent sinter.

In other words, the Dwight & Lloyd sintering-machine is a piece of chemical apparatus in which to carry out the reaction between the oxygen of the continuous supply of air and the solid combustible material of the charge. The former is maintained in a zone of regular and continuous supply moving in the direction of gravity. The latter is, of course, intimately mixed with the material to be sintered. It logically follows that the capacity of the machine is limited by the weight of carbon burned per unit of time. It has been found that an amount of carbon equivalent to 8 per cent. of the weight of the ore to be sintered furnishes ample heat for the production of the sinter. Carbon in the mixture in excess of 8 per cent. performs the wasteful and deleterious functions of (1) consuming proportionately longer time for its combustion; (2) raising the temperature of the sinter produced to its fusing-point, thus enveloping unconsumed carbon; and (3) raising the temperature of the grate-bars and other parts of the machine. Besides which, the minimum consumption of carbon, consistent with maximum production and most desirable quality of product, is desirable for economic reasons.

From a given material, properly mixed with the correct percentage of fuel, the character of the sinter produced depends wholly upon the material itself, being unaffected by the speed of sintering or velocity of air-supply; but its amount per unit of time depends upon the rate of combustion of the intermixed fuel, which, in turn, depends upon the rate at which the oxygen is brought into contact with fresh fuel, as the inert products of combustion are taken away by the action of the fan.

Thus we come to the principle that the permeability of the bed of material, and the pressure-drop between the atmosphere

outside and that in the wind-box, determine—other conditions remaining constant—the rate of the production of sinter.

Permeability is largely influenced by the moisture added. The function of the water is not entirely understood. Primarily it causes a temporary agglomeration of the particles of the materials, thus preventing them from passing through the grates. The voids in the bed are then larger, offering less resistance to the gas-current. It must be borne in mind that the amount of added moisture is not a given percentage by weight, but the quantity required to produce a certain degree of plasticity. For instance, to a fine hematite there could be added 15 per cent. by weight of water, before the material would cohere properly; but with magnetite, 4 per cent. of added water produces the equivalent degree of plasticity. The regulation of this phase of the process, which is doubtless very important, has as yet been determined by rule-of-thumb only. This is done very well by ordinary workmen after a small amount of practice, especially upon a given material.

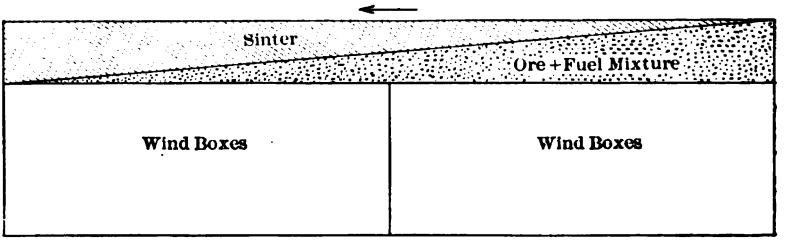

FIG. 1.—DIAGRAM SHOWING ARRANGEMENT OF MATERIAL IN THE ACTIVE ZONE—*i. e.*, THE PALLETS OVER THE WIND-BOXES.

The diagram, Fig. 1, shows the relative arrangement of the product and crude material in the active sintering-zone. The rectangle represents a vertical cross-section through the pallets over the wind-box. The sinter and ore-mixture are seen to be separated by an imaginary line beginning and ending at two respective zero-points. This is about the state in case of sintering any material. When high-carbon materials are sintering, the combustion of the carbon proceeds downward, progressively. Should high carbon follow low carbon directly, the only adjustment necessary is to slow down the speed of the pallets, allowing greater time for burning out the excess of carbon. The only condition under which residual carbon is left in

FIG. 2.—HEATING-CINDER.

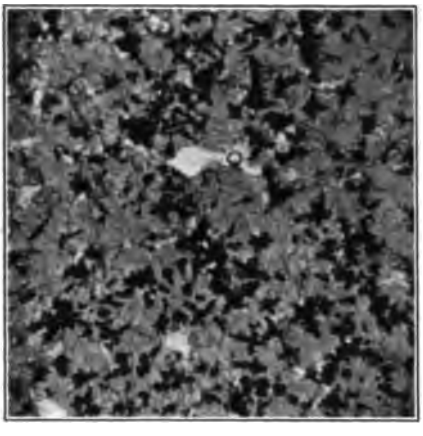

FIG. 3.—PUDDLE-CINDER.

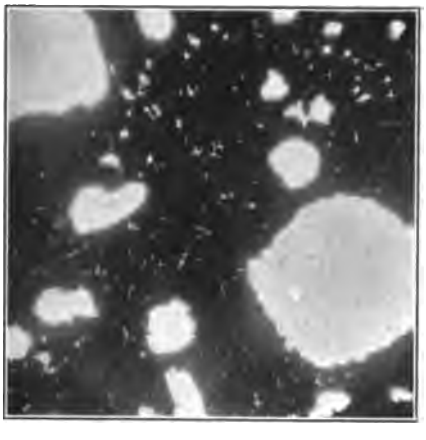

FIG. 4.—DWIGHT & LLOYD SINTER FROM FLUE-DUST.

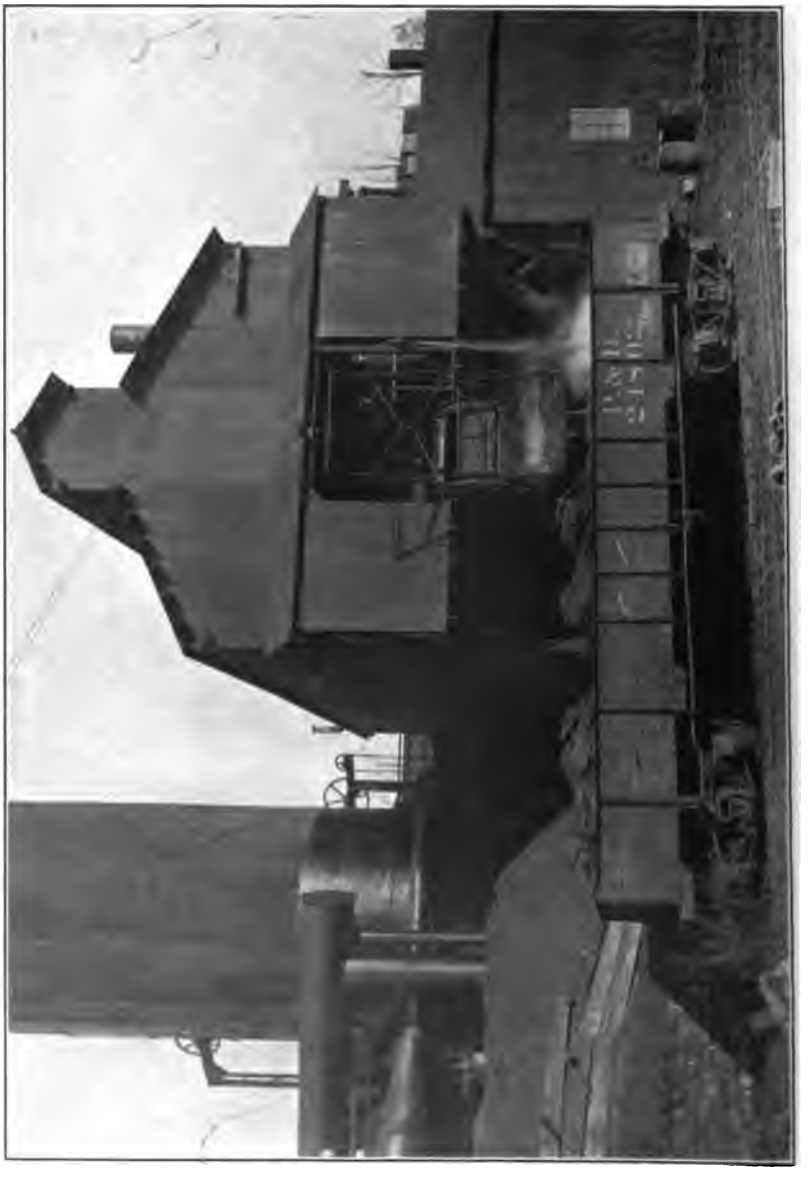

FIG. 5.—VIEW OF DISCHARGE-END OF FLUE-DUST CONVEYOR AND FINE-ORE SINTERING-MACHINE.

FIG. 6.—PALLETS CONTAINING SINTERED MATERIAL.

FIG. 7.—VIEW OF BLAST-FURNACE PLANT OF E. & G. BROOKE IRON CO., BIRDSBORO, PA., AND SINTERING-PLANT, SHOWING RELATIVE SIZE.

FIG. 8.—TOP OF ASSEMBLED SINTERING-MACHINE BEFORE COMPLETION OF BUILDING.

the sinter is when the temperature of the sinter in the upper part of the bed is raised, so as to render it incipiently fluid, and permit it to envelop small amounts of carbon. This only occurs when the carbon is excessively high, say, above 20 per cent.

In Mr. Gayley's article,[3] the tests showing the extent of desulphurization in sintering were noted. These were all made upon small samples, and belonged to the first trial of a particular material. There have been practically no high-sulphur materials treated which were not brought down as low in sulphur as the best Lake ores. No special adjustment or preparation of the ores is necessary to effect desulphurization; the sulphur is eliminated simultaneously with the sintering. The desulphurization is not dependent upon the form or quality of the ore. Pyrites cinder has been brought down from 4.41 to 0.07 per cent., while magnetites have been lowered in sulphur from 3.50 to 0.15 per cent.

An analogy between the sinter made by the Dwight & Lloyd process and that of mill-cinder has been suggested. No such comparison is practicable. The two materials are wholly different in origin, appearance, and ultimate structure. Whenever mill-cinder or puddle-cinder has a small amount of entrapped gases, producing pores, the walls of the pores are glazed, and are therefore impervious to gases. In fact, the material is a glass, which has been completely liquid. The Dwight & Lloyd sinter, made under normal conditions, has never approached the liquid state, and hence its particles are united by plastic coherence of their surfaces; and the cell-walls have a microscopic porosity exceeding that of most easily reducible ores. The micro-photographs shown in Figs. 2, 3, and 4 very forcibly show the distinction among the structures of the cell-walls of the different materials.

Fig. 2 shows a section of heating-cinder; Fig. 3, puddle-cinder; and Fig. 4, the Dwight & Lloyd sinter, all of which are under a magnification of 40 diameters. The first two materials have been cooled from a liquid state. By its method of formation the heating-cinder approaches more nearly a theoretical ferro-silicate. The puddle-cinder, although containing practically the same chemical compound as the heating-cinder, has a different proportion of the same constituents. The

[3] *Iron Age*, vol. lxxxix., No. 1, p. 73 (Jan. 4, 1912.)

heating-cinder is further subjected to slower cooling and hence is allowed more opportunity for perfect formation of the crystals. The translucent and glass-like structure of the iron silicate is shown in each case entirely enveloping the excess of the oxides of iron, which is shown opaque in each of the two cases. On the other hand, the section of Dwight & Lloyd sinter in Fig. 4 shows none of the translucent silicates whatever. But there is shown a uniform opaque mass, indicating the absence of the glassy, silicate formation, which is characteristic of the two types of mill-cinder.

The pores in the Dwight & Lloyd sinter are shown, all having the irregular lines forming the walls of the cells. When it is borne in mind that this section is magnified 40 diameters, the porosity of the structure as carried down into microscopic proportions, yet leaving sufficiently strong cell-walls, may be appreciated.

The practical furnace-manager may think it a long step from handling the tonnages necessary for average furnace-operation and such lengthy considerations of such minute structure, but all the furnace-reactions resolve themselves ultimately into microscopic proportions. Furthermore, this consideration is to show that there exists a positive distinction between the various forms of mill-cinder and Dwight & Lloyd sinter. However porous the mill-cinder may be, it can never approach in reducibility, the product under consideration.

Mill-cinder is a satisfied chemical compound, only giving up its oxygen at high temperatures, and then affording a minimum area of contact for reaction of reducing gases. Dwight & Lloyd sinter is a physical agglomeration of material treated, being bonded of particles of original size.

Several thousand tons of sinter have been produced at the Birdsboro plant and used in the blast-furnace. Figs. 5, 6, 7 and 8 are views of the furnaces and the sintering-machines at Birdsboro. At all times the sinter has shown itself a superior and beneficial material of the blast-furnace charge. By reason of variations in the ore-mixtures, no actual figures of the fuel-saving effected through the use of the sinter are available; but several times, as much as 10 per cent. of easily reducible Lake ores has been replaced by sinter; and the management unhesitatingly declares that taking off the ore and putting on the

sinter has the same influence on the fuel-consumption as would result from taking off entirely that amount of ore-burden.

At no time has there been any reaction in the furnace resembling that produced by mill-cinder. When mill-cinder is used with a burden of a high percentage of magnetic ores, a scouring action is often noted—especially when the mill-cinder is put on to the amount of 12 per cent. in one mass. There has never been at the Birdsboro furnace any scouring action or descent of unreduced oxides into the hearth since this sinter began to be used. According to all practical indications and theoretical considerations, there seems no doubt that a complete burden of Dwight & Lloyd sinter might be used advantageously.

The iron blast-furnace stands on a high plane as regards metallurgical efficiency, but there are yet vast opportunities for improvement. Not the least of these lies in the due preparation of ores and fluxes prior to charging.

Mr. Gayley, in his Institute paper, quoted from Schinz's book, *The Action of the Blast-Furnace* (1871), the following sentence:

"A chemical action can only take place between two bodies, however great their affinity, if they are in intimate contact with each other; and the rapidity of this action will be so much greater, the more numerous the points of contact are."

Mr. Gayley's purpose in this quotation was to lay stress upon the advantage of the Dwight & Lloyd sinter over other materials in giving a greater area of contact to the gases in the reducing-zone. But the importance of this law is equal if not greater in its application to the reactions between solids in the fusion-zone. And this is where blast-furnace practice seems farthest from the ideal at present. Materials are charged in groups of tons, which can only perform their proper functions by combinations between minute particles. It then seems only logical to say that, if greater intimacy of arrangement were provided between the bodies forming slag, a distinct advantage would be gained, in that a slag of the desired chemical composition would be continuously formed in the zone of fusion. In present practice, there is an intermittent production—first of a basic, and then of an acid slag (or *vice versa*)—all finding their way into the hearth, and making up a final fair average, but certainly not constituting, in the strict sense of the term, a continuous chemical operation.

The natural consequence of submitting to zones of varying temperature mixtures of varying melting-points is a discontinuous and disproportionately wide zone of fusion. This source of irregularity is probably the cause of many slips, scaffolds and other disturbances in furnace-practice.

As a possible remedy for these conditions, I would suggest a wider application of the Dwight & Lloyd sintering process. Let us assume, as a hypothetical case, a mixture containing all-fine ores of any specific burden (say 40 per cent. of the entire ore-charge), and the flux for the entire burden (crushed to pass 10-mesh and under), to which the proper amount of carbon is added and sinter is produced. This sinter now contains: (1) the iron partly metallic and partly as either oxide; (2) the gangue and flux, not necessarily combined, but intimately associated and free from all CO_2. While the lime is thoroughly calcined, it differs radically from a separately calcined lime, in that it is so agglomerated with other materials as to remain solid, instead of slaking and disintegrating.

The advantages to be expected from the use of such material in the blast-furnace are:

A.—The solid mixture, containing uniform slag-constituents, will carry them unseparated to the bosh, where their function will be performed continuously, instead of interruptedly (by "running ahead" and segregating).

B.—All the CO_2 having been eliminated, the solution of carbon by ascending CO_2 will be avoided.

Hence, from the reactions,

$$CaCO_3 = CaO + CO_2; \; CO_2 + C = 2\,CO,$$

there will be effected a saving of carbon equivalent to 12 per cent. of the weight of the stone. Assuming conditions of 1,150 lb. of limestone per ton of pig; 96 per cent. of $CaCO_3$ in limestone; 85 per cent. of available carbon in coke; then:

1,150 lb. of limestone $= 1{,}150 \times 0.96 = 1{,}104$ lb. of $CaCO_3$.

$1{,}104 \times 0.12 = 132$ lb. of carbon saved.

$132 \div 0.85 = 155$ lb. of coke per ton of pig saved.

C.—An increased reducibility of the iron so treated. (This has been conservatively estimated as 5 per cent. better than the best available ore.)

D.—As a larger portion of the slag-forming constituents will

have already been combined, an amount of carbon representing the difference between the heat for fusion and the heat of formation will be saved.

E.—Uniform delivery of properly-proportioned materials to the different zones.

There has been set a numerical value for only the first case, as it is almost universally conceded that the carbon-solution factor is equal to the theoretical. Upon presenting this proposition to a well-versed student of blast-furnace conditions he said, "the furnace which does that will double its output." I do not claim such sanguine advantages as this, but I feel that an advantage would be gained far in excess of the expense of sintering the materials.

Materials otherwise regarded as unfit for the blast-furnace may be readily and cheaply converted into a product most desirable for that purpose. For instance, as considerable flue-dust carries an amount of carbon in excess of that required for sintering its own iron-bearing constituents, this may be used by mixing the flue-dust with blue billy or magnetites, thus giving a product of uniform, cellular composition without cost for fuel.

The Dwight & Lloyd sintering process thus offers to the problem of the conservation of our mineral resources a solution applicable to present economical and industrial conditions.

Agglomeration of Fine Materials.

BY WALTER S. LANDIS, SOUTH BETHLEHEM, PA.

(New York Meeting, February, 1912.)

THE earliest example of attempting to form finely-divided materials into larger masses for better adaptation to commercial use was probably the briquetting of peat and lignite-waste at Paris by the use of a clay binder. It was from this attempt that our word briquette has arisen (Fr. *la brique*), the formed masses being shaped similar to ordinary bricks. This term does not, however, lend itself to the many shapes of such formed material as are now being produced, as few of them, aside from some of the European brown-coal products, bear any resemblance to the shape of that well-known article. The

term "agglomeration" has therefore been chosen as more accurately descriptive of the products now being manufactured, this term including the molding as well as the sintering.

With the increasing prices of fuel and ores and the greater demand for economy in the operation of industrial plants, more attention has been paid in recent years to the utilization of waste products and low-grade ores and fuels. Much of the waste has been occasioned by the inability to utilize finely-divided ores and fuels in the furnaces because they could not be kept there, or else because they clogged up the furnace-shaft so that gases could not be forced through under ordinary conditions of operation. As a result, all such fines were thrown on the dump. Again, with the exhaustion of the richer deposits of ores, concentration of lower-grade ores becomes a necessity, and most concentrates are produced in a finely-divided condition. Aside from these considerations, there is also the economy in the cost of operating the plant; a furnace running on agglomerated material will have a much greater output than one working on fines. In one instance the charge of an iron blast-furnace was changed from fine concentrates to briquettes and the output was increased four-fold; better results could have been obtained had facilities been at hand for taking care of the increased output.

To attempt to catalogue all the materials that have been agglomerated by one or another process is manifestly impracticable, and I give here a list of only a few of the more important ones at present attracting attention:

I. *Fuels.*—In the mining and preparation of fuels like coal and lignite much fine material is produced. Also, certain lignites slack and fall to pieces on storage. The agglomeration of these fines has long been practiced in Europe, where fuel is scarce and expensive, and the U. S. Geological Survey and the Bureau of Mines have been active in calling attention to this phase of conservation of our own fuel-supplies.

II. *Concentrates.*—The shaft-furnace, when fed with lump material, is probably the most economical of our smelting-devices. Also, the economy incident to the smelting of pure ores in such a furnace is too well known to merit discussion here. This has led to a wide application of ore-dressing to prepare such pure material; and since nearly all dressing-operations require more

or less fine crushing of the ore, it is almost a necessity that some agglomerating-device be used to prepare the concentrates for advantageous use in the blast-furnace. In the United States the agglomeration of magnetite concentrates has received considerable application.

III. *Fine Ore.*—Years ago many of our ores, particularly those of iron, were plentiful and cheap. Furnace-men would not accept fines, and as a consequence large quantities of this material accumulated at the mines and the furnaces. With the increasing cost of ore the heaps of fines all have increased in value and may be profitably agglomerated and sold. Moreover, the long hauls and frequent trans-shipments necessary with many ores in the United States cause much breakage, and this condition opens up another field for the industry.

IV. *Flue-Dust, Sweepings, Etc.*—The disposal of the dust carried out of blast-furnaces has frequently offered a problem quite in keeping with the actual intrinsic cost of the dust to be handled. The agglomeration of this material, together with the sweepings of the plants, is now attracting the attention of furnace-operators, and great saving would be attained by again charging such material back into the furnace in such form that it will not be blown out. In the year 1910 the United States produced nearly 28,000,000 tons of pig-iron. A conservative estimate of the flue-dust made by the blast-furnaces in the same year is 3,500,000 tons, carrying at least 35 per cent. of iron. At a low value of 4 cents per unit of iron, this dust cost almost $5,000,000 for its iron-content alone, not to mention another $1,500,000 worth of coke contained in it. Surely a process that would enable one to utilize this enormous by-product is worth attention.

V. *Scrap-Metal.*—Even finely-divided scrap-metal, such as filings, chips, etc., is now being formed into briquettes for more efficient handling. Sometimes flue-dust, sweepings, etc., are agglomerated together with the metal, the whole making a mass possessing many advantages in certain branches of metallurgical treatment.

No single method of procedure is applicable to the treatment of the widely-different materials listed in the above five classes. In fact, the diversity of the agglomeration processes used makes it somewhat difficult to outline a simple classification

which will be comprehensive, and the classification here proposed may sacrifice completeness for simplicity.

In general, the agglomeration processes may be divided into:

I. Those utilizing certain properties in the material for producing the desired coherence.

II. Those obtaining coherence through the addition of a foreign substance or binder.

Under Group I. are four classes:

1. Certain materials, when moistened with water and molded into form and dried, possess considerable coherence. Ores carrying soluble salts, clay, easily-hydrated compounds, etc., are frequently found amenable to this simple treatment. A few iron-ores and roasted products are actually treated in this manner, but the product possesses the great disadvantage that on heating to a temperature sufficiently high to drive off the water the briquettes fall to pieces.

2. Pressure greatly assists in developing the cohering-power of many materials; probably if a high enough pressure could be uniformly applied throughout the mass to be formed all substances would agglomerate under this treatment, but the use of pressure alone is limited to but very few cases in practice. Sometimes the best results are obtained on dry materials, sometimes a certain amount of moisture is necessary; the pressure so far used runs up to about 2,000 atmospheres.

3. All materials on being heated to a high enough temperature pass through a pasty or semi-fused state and cohere to such a degree that on cooling a more or less firm mass results. This phenomenon is called sintering. If done in a rotating furnace or rotary kiln the sticky masses roll together like snowballs, and small round particles are formed, which are called "nodules;" the process is called "nodulizing."

This sintering property of a material may be taken advantage of by molding the material into briquettes, using only a slight pressure to insure filling the mold, then burning the briquette so formed at a temperature that will insure cohesion. This is the principle of the well-known Gröndal process.

4. This class is represented by processes employing a combination of pressure and sintering. It is difficult to fix exactly the limits which differentiate a process truly belonging to this fourth class from one of the third, since the difference is wholly

in the degree of pressure employed. I have chosen arbitrarily to place in this fourth class all processes which use pressures exceeding 30 atmospheres for the formation of the briquette. The best example of this fourth class is the originally Ronay process as installed at Catasauqua, Pa., in 1904. The Ronay process has since been modified by limiting its application to ores which will agglomerate under high pressure alone, without the necessity of afterwards sintering them.

Under Group II. are included the greatest developments of the briquetting processes. The use of a binder much simplifies the whole operation, as the binder acts as a cementing medium to hold the inert particles of material together. It is almost impossible to list all the binders that have been used; a few of them are, clay, lime, ground slags, natural and Portland cements, water-glass, kieselguhr, carnallite, tar, pitch, asphalt, petroleum, sulphite residues, naphthaline, paraffine, molasses, resin, starch. Most of these act of themselves if the precaution is taken to insure a thorough mixing with the material to be briquetted. Others require that the briquettes be aged, or even heated after formation to temperatures up even to redness.

The operations of Group II. may therefore be divided into three sub-classes: (*a*) Processes in which the binder is mixed with fines and molded under low pressures; (*b*) processes in which higher pressures are used along with the binder; (*c*) processes in which a binder, and possibly pressure, are used, with a subsequent heat-treatment. The first two are self-explanatory; the last may be sub-divided into processes which use a baking in superheated steam, or even agglomerating in a furnace, as, for instance, when lime is added and later the briquettes are passed through an agglomerating-furnace.

It seems to me that proper attention has not always been paid to the selection of a binder for the particular case at hand. Too often is the intrinsic cost of the binder itself the main item in its selection, the subsequent costs it entails in passing through the smelting- or refining-furnace being entirely overlooked. The waste-product possessing binding-power may become very costly if it entails the production of an extra amount of slag to be smelted, or introduces in the furnace-product a small amount of impurity which is difficult to remove afterwards. Where

limestone is an essential ingredient of the furnace-charge lime can be profitably used in considerable quantities as a binder. If the materials are self-fluxing a minimum of binder should be used, no matter what its composition may be.

Pressure increases the effectiveness of all binders, enabling one to obtain the desired results with the use of a minimum of extraneous material. Around metallurgical works, where power is usually cheap, an extensive investigation of the most economical balance between binder and pressure should be carried out. The importance of the industry is such that soon some enterprising briquetting company will erect a central testing-plant similar to an ore-testing laboratory, and such questions as arise will be absolutely settled for each particular case at a minimum of expense.

The physical properties which a briquette should have may be summed up as follows:

It should be strong enough to stand handling without undue breakage. The simplest test to determine this property is a drop-test. A number of briquettes should be dropped from a height of at least 6 ft. upon a hard stone surface. Not over 25 per cent. of the number so dropped should break, and none of the broken ones should go to powder of the original texture of the material before briquetting. The breaking of a briquette into two or three pieces is not a serious matter, but in no case should it pulverize.

Where the agglomerated material is to be stored or transported before use a weathering-test should be applied. Storage in the open which subjects the agglomerated material to rain, snow, frost, and sunshine should be resorted to, since none of these agencies should affect a well-made product.

For proper reduction in the shaft-furnace the agglomerated material should be porous. Where briquettes are to be tested, the dropping of water from a measuring-device upon a briquette of known volume gives a fair test, if care be taken to do it slowly and carefully, so that no more water is measured out than is absorbed by the mass. A simpler and easier test is made by weighing the agglomerated mass and submerging it under water for at least 24 hr. and again weighing. The displacement of the entrapped air may be hastened by performing the operation under a vacuum. A good briquette should show a

porosity of 20 per cent.; that is, it should take up water to the extent of 20 per cent. of its volume. One of the greatest drawbacks to the nodulizing process is that the nodules are non-porous, and therefore difficult to reduce. The addition of lime to the charge of the nodulizing-kiln somewhat counteracts this tendency to form non-porous nodules, but unless carefully made the nodules will not withstand some of the other tests recommended.

Disintegration should not follow the exposure to water-vapor at 150° C. (300° F.), because the top of a blast-furnace contains vapor at this temperature. The test for disintegration is best performed by submitting the briquette to a steam-pressure of 60 lb. per sq. in. for several hours. Care must be taken to submit briquettes containing certain binders to this test, since some of them will fall to pieces under the action of hot water-vapor.

Briquettes should not disintegrate when heated to redness. This test can best be done in a muffle, and if conditions permit should be in an atmosphere of carbon monoxide and dioxide. Many of the hydrated briquettes, or those bound together by hydrating influences, will not stand this test.

Of course, it is understood that no briquette should contain a binder that exerts a harmful influence on the product of the furnace. No sulphur-compound, for instance, should ever be introduced into a briquette to go into an iron-furnace.

Sintering and Briquetting of Flue-Dust.

BY FELIX A. VOGEL, NEW YORK, N. Y.

(New York Meeting, February, 1912.)

FLUE-DUST, to most blast-furnace operators, means a troublesome by-product, the formation of which should be curtailed, if not prevented entirely. However, with the increasing use of fine ores, larger furnaces, and high-pressure blast, the production of flue-dust is constantly increasing, and amounts annually in the United States to from 3,000,000 to 3,500,000 tons, an exceedingly high tonnage, of which a large part has been discarded as valueless.

As a result of greater economy in the iron industry, the attention of our furnace-men has been directed towards the utilization of this enormous amount of waste material, a problem which had also been given due consideration by metallurgists abroad.

Flue-dust is generally a fine material containing considerable coke and iron-ore, with a small admixture of lime and silica, depending upon the burden. The iron-ore is partly reduced, which shows that the dust originates largely in the reducing-zone of the blast-furnace.

In the United States this dust usually contains 20 per cent. of coke and more than 40 per cent. of iron. Estimating coke to be worth $3.25 per ton, and iron-ore 7 cents per unit, a ton of flue-dust, unless made available, represents a loss to the furnace-man of $3.50. This accounts for the first efforts to recharge the flue-dust into the furnace, either by moistening it down with an excess of water, or mixing it with clay to form balls of pulp, or treating it with lime-water.

These methods, however, have been practically discarded, as they failed to produce the desired economies.

To recover, in the blast-furnace, all the values represented by the material contained in the flue-dust, the following conditions should be complied with:

1. The dust should be agglomerated into lumps about the size of furnace-coke, so that it will help to carry the burden and facilitate the flow of the gases.

2. The agglomerated material should be strong enough to carry the burden without disintegrating; it should be heavy, so as to decrease its volume, and it should be sufficiently porous to permit the furnace-gases to penetrate fully. Under no condition should the surface be glazed.

3. It should contain all of the valuable constituents of the dust, such as coke, iron-ore, lime, etc.

4. It must stand handling, without undue breakage, and should not produce more than 5 per cent. of dust.

5. It must stand the weather.

6. It must not disintegrate in the blast-furnace before being greatly or totally reduced.

7. It must submit to easy reduction without requiring additional fuel.

8. It must not contain substances detrimental to blast-furnace operations.

9. Its cost of production must be low.

Blast-furnace operations, by the use of such agglomerated material, will result in

1. Regular steady operation.
2. Increased burden, with increase in the metal produced.
3. Decrease in the consumption of coke.
4. Decrease in the production of flue-dust.
5. Decrease in the cost of producing pig-iron.

In order to obtain these results American and European metallurgists have followed two different lines. In the United States it has sufficed to save the iron-content only (though not in the best possible form), while abroad the endeavor has been to make a high-class product which would meet all the conditions above enumerated.

A number of processes have been evolved in the United States, generally known as agglomerating and sintering processes. They are based on fritting together the particles of ore by heat, the binding action being due to the formation of silicates, mostly of iron. When the coke has not been eliminated mechanically from the dust, it is burned out, leaving ash in the agglomerated material, which increases the formation of iron silicates or glazing-material. In some cases, however, more fuel is added to the flue-dust, which, naturally, further increases this drawback.

The nodulizing process, the oldest process in the United States, has been in successful operation for a number of years.

Flue-dust is either treated directly, or it is previously submitted to a magnetic separation to eliminate coke and lime. This is done to facilitate the subsequent nodulizing-operation, which is carried on in a slowly-rotating cement-kiln from 80 to 120 ft. long. From 200 to 300 lb. of finely-powdered coal is used per ton of finished material, the coal being blown into the kiln. Gas and oil have also been tried with more or less success.

The heat produced is considerable and difficult to control, the semi-soft material formed consisting of iron-ore particles and slag, which, by the revolving action of the kiln, is balled together in nodules of various sizes—from a pinhead to that of

a cannon-ball—which are usually quite dense, often fused and glazed. They contain from 60 to 67 per cent. of iron, which makes them quite attractive from the furnace-man's point of view.

To make nodules an elaborate and expensive plant is required, the operation of which is more or less difficult and costly.

The Huntington-Heberlein pot process, which has been used with good results in the roasting of pyrite-cinders, has been recommended for the fritting of flue-dust. The resulting fritted material is of more or less cellular structure. However, this would not be an advantage, as the surface would be largely glazed, rendering it not permeable to gases, and would have to be removed in the blast-furnace at about smelting-heat.

The process is somewhat simple, requiring stationary iron pots fitted with a perforated false bottom through which the air is blown into the charge. The equipment is cheap; the operation, however, is not continuous, which makes it expensive.

The Gröndal briquetting process has also been applied to flue-dust. The flue-dust, either moistened, or after the elimination of the coke and stone, is pressed into bricks, which are subsequently fritted in high temperature. To facilitate the operation it was found necessary to eliminate the coke; this, however, increases the cost. The separated coke is of little value, since it contains many impurities. The presses used are ordinary brick-presses. The bricks are carefully placed in layers on cars, which are run into kilns about 170 ft. long, where they are submitted to high heat, gas being used for the purpose. The platforms of the cars are built of fire-brick and form the bottom of the kiln. The highest temperature, about 1,300° or 1,400° C., is reached in the center of the kiln, the highly-burned briquettes working gradually towards the cooler end of the kiln, where they are finally unloaded. The operation lasts about 7 hr. The resultant briquettes contain from 65 to 70 per cent. of iron with practically no impurities, these having been eliminated. No ashes are left by the fuel. The briquettes are only slightly fritted, are very porous and friable, and make a high-class material for use in an open-hearth furnace.

The manufacture of Gröndal briquettes is expensive, necessitating a costly plant, which has limited output.

A ton of flue-dust will produce about two-thirds of its weight in briquettes.

Quite recently the Dwight-Lloyd process has been applied to the sintering of flue-dust. The material is submitted to internal combustion in layers from 5 to 7 in. thick. It is fed on an endless conveyor formed of iron pallets or grates. After the fuel contained in the flue-dust has been ignited by means of a gasoline torch, or some other device, the air is drawn through by suction. The operation lasts about 20 minutes.

A good deal has been published of late in regard to this process. It is claimed that the required plant is not costly, while the operating-expenses are considerably lower than in the previous process. The resultant sintered material is not homogeneous, and, while a large portion of it is of cellular structure, it is glazed on the surface, which makes it quite difficult to reduce in the blast-furnace.

The Greenawalt process uses much the same apparatus as the Heberlein pot process, but air is drawn through the charge as in the Dwight-Lloyd process.

The sintered material from these various processes is expensive on account of the loss of carbon, and, with the exception of the Gröndal briquette, is not of good physical structure and is usually glazed on the surface.

The briquetting of flue-dust has been more attractive abroad than has the sintering.

The lime process mixes the flue-dust with from 5 to 10 per cent. of calcium hydrate. After briquetting, the material is exposed to the air for a certain length of time, so that a carbonate is formed, which is the binding medium. This binder will eventually act as flux and replace a certain amount of stone. The process, while having decided advantages, is quite cumbersome and costly, as the briquettes must dry from two to four weeks under cover.

The Pioneer process uses sulphite pitch (obtained from the sulphite pulp-mills) as a binder. It is an organic substance, rich in carbon and hydrocarbons, which will burn readily and thus increase the caloric value of the material. The flue-dust is pressed into briquettes with from 4 to 8 per cent. of sulphite

pitch; they are quite hard and give fair results, but the process is expensive.

The Ronay process does not use a binder. The flue-dust is submitted to a very high pressure in a specially-constructed type of hydraulic press; the resulting briquettes can be handled immediately and have proved very satisfactory. The process requires an expensive plant, however, increasing the operating-cost.

The Schumacher process does not use what may properly be called a binder, but is based on the latent cementing actions existing in fresh flue-dust and which are made active by the presence of a small amount of a catalytic substance. Thus 0.25 per cent. of magnesium chloride mixed with fresh flue-dust and from 6 to 10 per cent. of water, pressed into briquettes, will create a strong reaction, noticeable by the considerable heat developed; the briquettes will be perfectly set and hard within a few hours. The process is very simple, an ordinary pug-mill being used in which to mix the material, which is subsequently pressed into briquettes in a toggle-press and then loaded on cars to allow them to set.

Some flue-dust will react so strongly that a large amount of ore or coke-breeze may be added to the briquettes; in these cases the flue-dust acts as a binder.

The Schumacher process requires an inexpensive plant and the cost of manufacturing is low. It is extensively used abroad and has replaced some of the other methods. The briquettes of all agglomerated materials have given most satisfactory results abroad and highest returns and values to the blast-furnace operators.

The Schumacher Briquetting Process.

BY JOSEPH W. RICHARDS, SOUTH BETHLEHEM, PA.

(New York Meeting, February, 1912.)

THIS method of briquetting flue-dust, or flue-dust mixed with fine ores, or, in a few exceptional cases, coke-dust, has come into large commercial use in Europe, and a small plant is already in operation in the United States. It promises to become of such great importance to the iron industry of the United States that the following description will certainly interest the majority of our pig-iron producers.

Discovered in 1908, the process rests upon the observation of Dr. Schumacher that blast-furnace flue-dust, while possessing of itself no binding or cementing properties, acquires strong cementing properties if mixed with a very small amount of certain salt solutions, the amount of which is so small that the resulting cementation appears to be rather by catalytic action of the salts added than by the mass-action of such chemicals. To be more specific, fresh blast-furnace flue-dust mixed with from 5 to 10 per cent. of its weight of magnesium chloride or calcium chloride solution—that is to say, with from 0.25 to 2 per cent. of its weight of magnesium chloride or calcium chloride—acquires the property of setting within a short time, from 15 to 60 min., and forming a hard cemented briquette. Alkaline solutions appear to have no action such as described, and they even interfere, if present, with the action of the magnesium and calcium solutions.

When treated in this way the flue-dust briquette sets very hard, apparently from pure excess of cementing- and setting-power. It is, therefore, possible to mix with the flue-dust a considerable proportion of inert ore or like material, which has no setting-power by this process, and thus to make a compound briquette containing large quantities of ore mixed with flue-dust, but in which the flue-dust may be regarded as the cement or binding-material. A particularly strongly cementing flue-

dust will carry as much as three times its weight of fine ore or the like, and yet produce a satisfactorily strong briquette.

The Cambria Steel Co. has installed at Johnstown, Pa., a small plant consisting of two presses, with a combined capacity of 250 tons of briquettes per day. One of the presses is of the mechanical type, with hydraulic safety-regulator; the other is a Ronay hydraulic press. The first makes briquettes the size of an ordinary fire-brick; the latter makes briquettes cylindrical in shape, 8 in. high and 8 in. in diameter, using considerably higher pressures than the former.

Fig. 1 is a view of a Brück-Kretschle press for the Schumacher process, installed at the works of the Société Cockerill, at Seraing, Belgium, which is similar to the one at Johnstown, and Fig. 2 shows the plant complete, from hoppers and mixer to discharge-belt.

As seen in operation early this month, the warm flue-dust from the dust-catchers was stored in one hopper, while fine Mahoning ore, containing from 10 to 12 per cent. of moisture, and quite cold, was stored in another hopper. A rotating feeding-device formed a stream of these materials, passing into the mixing-trough with ordinary spiral, into which at the same time was run a 30-per cent. solution of calcium chloride, the flow of which was regulated by a hand-valve. The trough fed the mixture directly into the hopper of the press. The flue-dust was warm, approximately at 90° C.; the ore was ice-cold; the solution was cold. The mixture fed to the press was just warm to the touch; the bricks going from the press were decidedly hot and steaming, and when loaded into the car they were quite hot to the touch and steamed vigorously.

This heating of the mixture as soon as compressed into the briquette is a characteristic of the process, and is an index of the rapid chemical reaction taking place which results in the cementing of the material.

The pressures used on the two presses seen were 5,500 lb. per square inch on the Brück-Kretschle press and 12,500 lb. per square inch on the Ronay press. The first press consumed 35 h-p., with an output of 7 tons per hour; the second press, 25 h-p., with an output of 4 tons per hour.

The flue-dust carries from 18 to 20 per cent. of coke, which is all carried into the mixture, said mixture being 70 of ore to

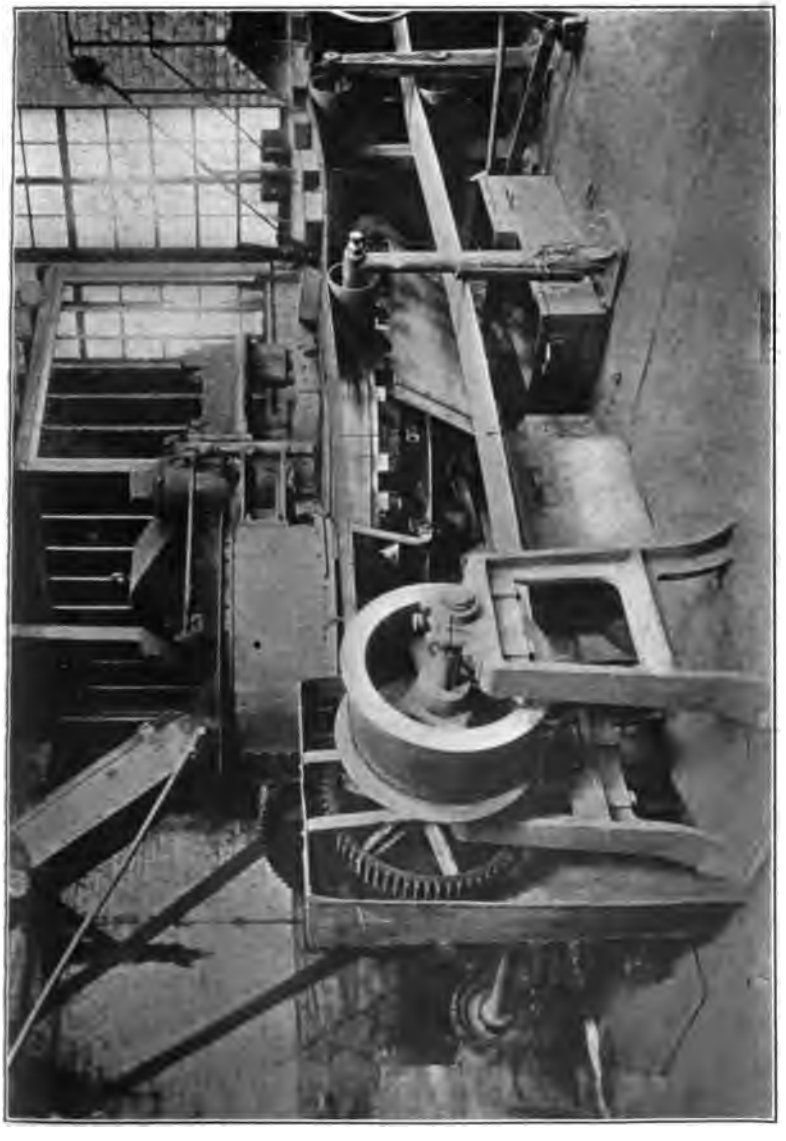

FIG. 1.—THE BRÜCK-KRETSCHLE MECHANICAL PRESS.

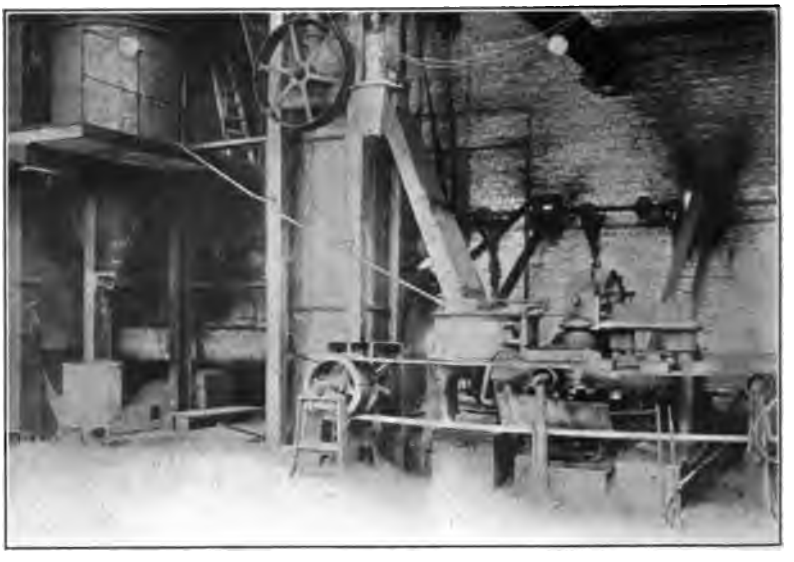

FIG. 2.—VIEW OF COMPLETE SCHUMACHER BRIQUETTING-PLANT.

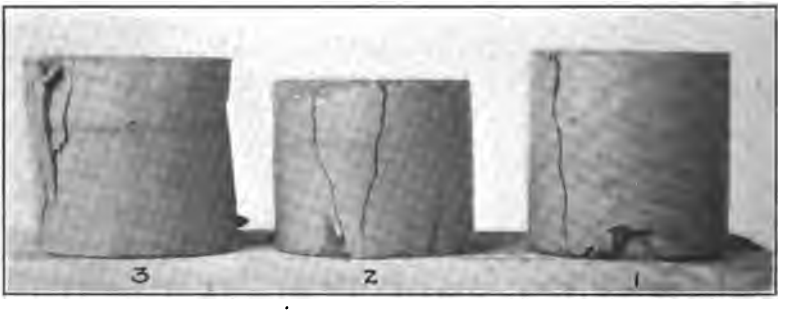

FIG. 3.—SCHUMACHER BRIQUETTES OF 70 ORE AND 30 FLUE-DUST, AFTER COMPRESSION-TEST.

30 of flue-dust. There is from 5.5 to 6 per cent. of coke in a finished briquette. The saving of this coke, 20 per cent. in weight of the flue-dust used, represents sufficient value to pay for the entire cost of the briquetting of the dust. Undoubtedly the coke thus inclosed in the briquette acts on the ore in the furnaces, and reduces it with at least as great an efficiency as the "green" coke put into the furnace. Therefore, if the furnace has sufficient smelting-power, other things being equal, this coke of the briquette may be estimated as saving at least an equal weight of coke to the furnace.

As to physical properties, a briquette made of flue-dust, tested by me, absorbed 11.5 per cent. of its weight of water, representing 27 per cent. of voids, a very satisfactory porosity.

Briquettes one day old, hard dried, tested for compressive strength by me, gave a minimum of 445 lb. per sq. in (31.8 kg. per sq. cm.). Photographs of three briquettes thus tested are shown in Fig. 3.

Several hundred tons of these briquettes were already in the ordinary furnace-bins, ready for use in the furnaces, and appeared to have stood transportation to that point and dropping from 10 to 20 ft., and also subsequent action of ice and snow for several weeks, without deterioration; only a few were broken, and but little dust was made.

The operation of producing briquettes in this manner, from flue-dust and fine ores, appears to have attained the maximum of simplicity, the operation consisting merely of mixing with solution and pressing into shape. Estimates of probable cost, if the operation were carried out upon a scale such as would suit a large blast-furnace plant, yielded as the outside figure 30 cents per ton, inclusive of depreciation, repairs, and interest on plant.

The inventor of this process has certainly made a great advance in the art of briquetting these difficult materials at a minimum cost and with maximum simplicity of plant. It is not difficult to predict a very large application of this process in the United States, as soon as the behavior of the briquettes in the blast-furnace has been proved. The very satisfactory running of similar briquettes in European blast-furnaces appears to leave no doubt upon this phase of the question also.

Theory of the Process.

Laboratory-experiments were made upon pure materials representing the constituents of flue-dust, with 0.5 per cent. of calcium chloride, with the following results:

Ground silica, after standing in air: Caked slightly under pressure with water alone; no perceptible difference with the chloride solution.

Ground silica, freshly burned: Same result.

Fine alumina, after standing in air: Same result.

Fine alumina, freshly burned: Same result.

Volatilized silica: Same result.

Coke-dust: Same result.

Pure ferric oxide, ignited at redness: With water alone, and pressed, very feebly coherent; with the calcium chloride solution, set hard and firm, and harder the greater the pressure.

Since the latter compound was chemically pure, we must find an explanation from the interaction of Fe_2O_3, $CaCl_2$ and H_2O, facilitated by pressure. The bottle of calcium chloride solution used showed a deposit of calcium hydrate on standing, caused by the well-known phenomenon of hydrolysis, and the solution became slightly acid. The reaction is

$$CaCl_2 + 2H_2O = Ca(OH)_2 + 2HCl.$$

This led us to suppose that the acid thus tending to form would immediately act on the iron oxide, forming ferric chloride, the reaction being:

$$Fe_2O_3 + 6HCl = 2FeCl_3 + 3H_2O.$$

But iron salts are precipitated by milk of lime (calcium hydrate), a well-known reaction extensively used in metallurgy and chemistry. It seems fair to suppose, then, that the ferric chloride will react with the calcium hydrate, the reaction being:

$$2FeCl_3 + 3Ca(OH)_2 = 2Fe(OH)_3 + 3CaCl_2.$$

This completes the cycle, giving back the calcium chloride with which we started, the net result being simply the conversion of Fe_2O_3 into precipitated $Fe(OH)_3$. The latter, being precipitated at the points of contact, where, under pressure,

all these reactions are taking place, forms the bonding material cementing the grains together.

The heat of hydration of Fe_2O_3 is not high, but it is positive, and would theoretically heat a lump of pure Fe_2O_3 about 80° C. This may, therefore, well be the source of the warming-up of the briquettes. It is not unlikely that the $CaCl_2$ present also ultimately sets as $CaCl_2.6H_2O$ crystals, still further strengthening the bonding action.

It has been objected to the above explanation that a briquette thus bonded would not stand heat, but would fall to powder again on being dehydrated. This did not appear probable, from a consideration of the characteristic shrinking together and hardening or fritting of $Fe(OH)_3$ on being ignited (all chemists know how this precipitate burns to a hard, horn-like mass); but, as Deville used to say, "*ce n'est pas nécessaire de théoriser si on peut faire l'expérience,*" and accordingly the briquette of pure iron oxide with the Schumacher solution was heated to redness: it stayed in a single lump and became harder than before.

It appears that the explanation of Dr. Schumacher, that the solution "acts as a catalytic agent to develop the latent setting-power of the flue-dust," is not very wide of the mark, and fits the facts, if we assume the reaction to proceed by the circular procedure (*Kreis Process*) above set forth, about as well as the term "catalytic" fits many chemical phenomena to which it is usually applied.

The Briquetting of Iron-Ores.

BY N. V. HANSELL, NEW YORK, N. Y.

I. Introduction.

The last few years have shown an increasing interest in the subject of beneficiating iron-ores in all iron-producing countries. In the United States, this movement has been slower than in certain parts of Europe, for the obvious reason that the abundance and relative cheapness of the Lake Superior iron-ores has hindered the development of enterprises for the preparation of low-grade ores. Gradually, however, conditions have changed. The world's consumption of iron is increasing enormously. Being in 1900 about 40,000,000 tons, in 1910 it was already 60,000,000; and there is no reason to doubt that for the near future the increase in iron-consumption will continue at the same rapid rate, especially with the development of China, certain parts of Africa, and other regions which have hardly been touched as yet by modern industrialism. It is, therefore, not surprising that apprehension is felt that the known available iron-ore resources of the world are gradually being depleted. Signs of this feeling are the recent investigation, under the direction of the U. S. Geological Survey, of the iron-ore resources of the United States, and the estimate presented to the last International Geological Congress, at Stockholm, Sweden, of the iron-ore resources of the world. These investigations have emphasized the fact that the known high-grade ore-deposits are limited in extent, and will, at a not distant future, be exhausted, compelling the iron industry to depend for raw material on the enormous deposits of low-grade ores which are distributed over almost all the world. The discovery of new deposits of high-grade ore, in countries not yet thoroughly explored, will not materially change this prospect. Of course, the part which such new deposits will play in the future cannot be forecast. The difficulty and cost of transportation will probably exclude many of them from use at the present centers of the iron industry.

In view of these facts, it is no wonder that great interest is displayed in the subject of making useful such ores as have heretofore been discarded because, in their natural state, they are not suitable for blast-furnace operations. First among these are ores the low iron-content of which precludes their direct use in the furnace, so that they have to be crushed and concentrated, giving as result a finely-granulated product, high in iron; and, in the second place, there are the ores which, although sufficiently high in iron, are rendered undesirable by either their physical or their chemical character.

Modern blast-furnaces, with their high stacks and heavy blast-pressure, make it difficult to include in the burden too high a percentage of dusty or finely-granulated ore, such, for instance, as some of the fine Mesabi ores. There are furnaces where conditions compel the management to use 60 or 70 per cent. of such ores; indeed, I am told of furnaces using 100 per cent.; but in such a case the manager will voluntarily tell his woeful tale of the unavoidable losses in the form of flue-dust, of scaffolding in the furnace, and of frequent explosions. The fine ore, descending more quickly than the rest of the charge, reaches the smelting-zone only partly reduced. This causes disturbance in the furnace-operations, with frequent casts of off-iron. Therefore, if it can be economically done, it is highly desirable to convert to a lump form all such fine ores, whether they have been obtained in the form of fine concentrates in the separation of low-grade iron-ores, or mined as natural soft and fine ores. A high sulphur-content in the fine ore is often an additional reason for its preparation by agglomeration, since the sulphur-content is usually reduced by the process, making the ore doubly attractive to blast-furnace managers.

In this country, however, the real impetus to the development of processes for agglomerating fine iron-ores, has been, not the desire to promote the utilization of such natural fine ores, but rather the necessity of finding some way of preparing for blast-furnace use such waste products as flue-dust, pyrites-residues, etc. The most important of these is the flue-dust, of which it is reported that between 2,000,000 and 3,000,000 tons are annually produced in the United States. A part of this is recharged into the furnace after a thorough wetting, intended to prevent its being directly blown out again. That this practice is not satis-

factory is indicated by the vast and growing piles of this material around furnace-plants in the Middle West, where a high percentage of soft ores is used in the burden. This waste is especially great when a furnace has been long in blast, and begins to be somewhat rough in the lining. The flue-dust contains from 35 to 45 per cent. of metallic iron, with from 5 to 20 per cent. of coke. Assuming the value of the original ore at the furnace as, say, 7 cents per unit of iron, the intrinsic value of this so-called waste material is probably about $3 per ton. Low iron-prices, and the consequent necessity of greater economy on the part of the blast-furnace manager, have brought to light this leak, through which often as much as 10 per cent. of the ore charged goes for naught.

Of the pyrites-cinder, about half a million tons are produced annually. On account of its usual pulverous condition and high sulphur-content, it was formerly thrown away, or perhaps, at the best, used for reclaiming land or building roads, although it frequently contained as much as 55 or 60 per cent. of iron.

The modern methods of utilizing for the iron industry, through agglomeration, these waste products or these low-grade ores, which in their natural state are unsuitable for blast-furnace operation, certainly form a not unimportant element in the present movement for the conservation of natural resources.

II. Methods of Agglomeration.

The many methods proposed or now employed for the agglomeration of fine iron-ores can properly be divided according to radical differences in the processes themselves, and also to some extent in the nature of the products, into the following three classes: First, nodulizing in revolving kilns; second, blast-roasting; and third, briquetting, with or without binding material, usually followed by a heating of the briquettes.

1. *Nodulizing.*

This method gained early prominence in the United States, where a number of plants are now in operation; the pioneer plant at Hackensack Meadows, N. J., treating pyrites-cinder; others, like those at Lebanon, Pa., Benson, N. Y., and Standish,

N. Y., treating ore-concentrates; and still others, like the Hubbard plant in Ohio, treating flue-dust. The most modern establishment of the kind is the one at Felton, Cuba, described by J. E. Little.[1]

In Germany, similar methods are in use for the roasting and sintering of carbonates; and in Canada there is a plant of four kilns now under erection for the same purpose. For such ores as decrepitate and shrink considerably in roasting, the method is specially valuable. The product is, however, seldom uniform. Occasionally, the nodules are vitrified and very dense; occasionally they are loose and brittle, producing much dust in handling. Difficulties in keeping the kilns free from scaffolds or rings, necessitating shut-downs every 8 or 10 days, tend to make the cost of production higher than a first investigation of the merits of the process seems to indicate.

2. *Blast-Roasting.*

It is not necessary to dwell at any length on this process, which has been described by James Gayley,[2] whose paper was presented orally by Arthur S. Dwight at the Wilkes-Barre meeting of the Institute, June, 1911, and by B. G. Klugh in a paper presented at this meeting. The product is spongy, and forms undoubtedly an excellent raw material for the blast-furnace.

3. *Briquetting.*

At a meeting of the Iron and Steel Institute in 1910, Chevalier C. De Schwarz, of Liège, read a paper on the briquetting of iron-ores.[3] In this he gave as conditions for successful briquetting:

"1. The iron ore briquettes must have a certain resistance against mechanical influences. They must resist a pressure of not less than 2000 lbs. per square inch, and, when dropped from a height of 10 feet on a cast-iron plate, they must not fall into dust although they may break into pieces.

"2. They must resist heat. Heated to 900° C. they may commence to sinter, but they must not disintegrate into small fragments.

"3. They should be capable of being placed in water for a certain time without softening.

"4. They must resist the influence of steam at 150° C. without crumbling.

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[1] *Trans.*, xlii., 152 (1911).

[2] *Trans.*, xlii., 180 (1911).

[3] *Journal of the Iron and Steel Institute*, vol. lxxxii., p. 10 (No. II., 1910).

"5. They must possess a certain amount of porosity in order to allow the carbon-monoxide in the blast-furnace to penetrate the interior of the briquette and to exercise its beneficial reducing influence."

"6. The binding medium, if any is used, should not contain noxious substances (sulphur, arsenic) to such an extent as to be injurious to the quality of the pig iron produced.

"7. The cost of producing briquettes should not exceed the difference in the prices between lump ore and fine ore."

In other words, the briquettes must be able to endure handling, transportation, and storing in the open air without too much disintegration, and to exist in the upper part of a blast-furnace without crumbling into dust. In addition, they ought to be easily reducible. If a binder is used, it should be of such a nature that it adds no deleterious elements; and it ought not to lower too much the iron-content of the material.

In calculating the technical economy, by which any process must stand or fall, it must be recognized that this cannot be determined by the briquetting-cost alone. The value of the finished product and its influence on the cost of pig-iron manufacture must be taken into consideration. For the preparation of a briquette which is easily reducible with a low consumption of coke, and the use of which increases the output of the blast-furnace, one can afford to pay more.

Of the numerous proposed methods, several of which have gained commercial prominence, some are better adapted to certain classes of ore than others. This is natural, since the processes have been originated and developed at places where necessity has called for them. Mr. Schwarz in his paper describes a briquetting process used at Kertsch, Russia, and Ilsede, Germany, where the ores are of a clayey nature. After adding from 6 to 8 per cent. of water to the ore, it is pressed into bricks at a low pressure (from 4,000 to 5,600 lb. per sq. in.), and heated to a temperature of only 75° C. The use of higher pressure would make the briquettes too dense. Parallel cases can be cited from other places. A high sulphur-content in the ore, for instance, gives prominence to a method by which in the briquetting the ore is heated sufficiently for an effective roasting.

To describe even the main features only of the various briquetting processes that have come into commercial use in Germany, for instance, would take me too far away from the

real subject of my paper. Persons interested may be referred to G. Francke.[4] But I will mention some of the more successful methods, in order to compare them with the Gröndal method, which it is my purpose, in this paper, to describe. The various briquetting processes may be divided into those that use a binder and those that do not. This is a very practical distinction. There can be little question that a binder used to cement together the ore-grains fills the voids between them, producing a briquette of low porosity. This must have an unfavorable influence on the behavior of the briquettes in a blast-furnace. The more porous an ore or a briquette, the more easily is it reduced. This is obvious, as in the upper part of the blast-furnace the ascending gases penetrate the porous briquette, and deposit fine carbon through dissociation throughout it, which causes a direct reduction before it has reached a great depth in the stack. The binder also lowers the iron-content of the briquette. In most cases from 8 to 10 per cent. of binder is employed; and the smelting and slagging of this material increases the fuel-consumption.

The binders most in use seem to be of the nature of hydrosilicates, which are hardened either by exposure in air a long time or by being treated with steam under pressure. The *Deutsche Brikettierungs Gesellschaft* advocates a method of this kind, forming with the binder a calcium silicate. The *Scoria Gesellschaft*, of Dortmund, uses from 8 to 10 per cent. of basic blast-furnace slag, or a mixture of 4 per cent. of slag with 4 per cent. of CaO. The ore is ground with the binder, pressed into briquettes, and hardened in steam. The Weiss method[5] is to mix the ore with 5 or 6 per cent. of hydrated lime, form the briquettes at a pressure of 4,500 lb., and expose them to the influence of CO_2 gases, first cold and then hot, at 300 lb. pressure. The Tigler process (Duisburg-Meiderich) employs a binder consisting of from 6 to 8 per cent. of slaked lime, sometimes with an addition of 1 per cent. of blast-furnace slag. The Schumacher process for flue-dust utilizes the hydraulic properties of the flue-dust, which, like cement, contains lime, alumina, and soluble silica ready for combination.

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[4] *Handbuch der Brikettbereitung* (Stuttgart, 1910).

[5] *Stahl und Eisen*, vol. xxxi., No. 38, p. 1539 (Sept. 22, 1911).

By the addition of a small amount of magnesium chloride, calcium chloride, or certain other salts, good briquettes can be obtained. The flue-dust ought to be used hot, direct from the dust-catchers.

Many organic binders have been tried, but few have been satisfactory. Trainer uses 4.5 per cent. of *zellpech*, which is a waste lye, resulting from the manufacture of sulphite pulp. This and similar binders coke in the upper part of the blast-furnace stack, and thus continue to hold the fine ore together until a reducing-zone in the furnace is reached. They are, however, generally too expensive to be used.

Of methods without binder, I will mention the one known under the name of Ronay. For the binder is substituted a high pressure (from 900 to 1,000 atmospheres) in hydraulic presses. This process has been more used for metal filings, etc., than for ores. The briquetting of clayey ores without a binder has already been mentioned. In reality, the clayey constituent of the ore serves as a binder.

The only process of importance that remains to be spoken of is the Gröndal, which consists in the briquetting of fine ore, etc., at a medium pressure, and the burning of the briquettes at a high temperature in a specially-constructed channel-furnace.

III. The Gröndal Process.

In 1899, when Dr. Gustav Gröndal was manager of the Pitkaranta Iron Works, in Finland, he built his first channel-furnace for the briquetting of iron-ore concentrates. Similar furnaces had been used previously for the burning of clay bricks, but for the adaptation of the process to the treatment of iron-ores and for the gradual development of a modified type of furnace suited to this special purpose the honor belongs to Dr. Gröndal. Figs. 1 and 2 are views of a pair of Gröndal furnaces, the latter showing the car loaded with briquettes. Sectional views of the furnace are presented in Fig. 3.

The first furnace at Pitkaranta was completely successful, and a good deal was written about it in the contemporary technical press of Europe. It was followed by one at Bredsjö, Sweden, built in 1902. Since that time, a great number of furnaces have been built both in Europe and in America.

FIG. 1.—A PAIR OF GRÖNDAL BRIQUETTING-FURNACES.

FIG. 2.—ONE END OF A GRÖNDAL FURNACE, SHOWING A CAR LOADED WITH BRIQUETTES.

At present, there are 16 plants in Sweden, with a total of about 35 furnaces and a briquetting-capacity of about 400,000 tons a year. In Norway, there are 3 large plants: Sydvaranger with 8, Salangen with 4, and Dunderland with 4 furnaces. In England 12 furnaces have been built, all treating pyrites-residues and "blue billy." Italy has 1 plant, Spain has 2; and in the United States there are 6 plants in operation or construction. In Canada the first one is being built by the Moose Mountain, Ltd., for the treatment of magnetite concentrates.

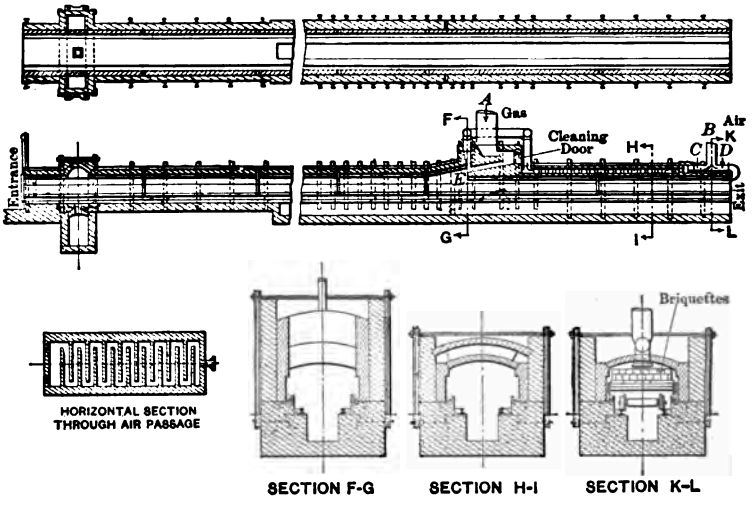

FIG. 3.—THE GRÖNDAL BRIQUETTING-FURNACE. SECTIONAL VIEWS.

1. *Construction of Furnace.*

The Gröndal furnace can briefly be described as a channel-furnace, through which the ore, previously pressed into bricks, is transported on flat cars, which form a continuous movable platform. It is heated by means of gas introduced through the raised arch at a distance from the entrance-end of about two-thirds of the furnace-length. The flat cars are usually built of structural steel with fire-brick tops, and are furnished on each side with flanges dipping into sand-troughs. They are often built with overlapping ends so that when they are pushed against each other in the furnace they separate completely the upper part of the furnace, through which the gases pass, and the lower part, in which are the trucks and the wheels. The furnace-walls are built double, with an outside

wall of red brick and an inside one of fire-brick; the two being separated by an air-space, which assists in lowering heat-losses by radiation, and at the same time permits the walls to move independently of each other in expanding and contracting on account of changes in temperature when the furnace is started and stopped.

For obtaining the necessary temperature, almost any kind of fuel that has sufficient calorific value can be used. In Europe, the rule has been to use producer-gas, except at some places in Sweden, where the furnaces are built at iron-works, and a mixture of blast-furnace gas and producer-gas is burnt. Blast-furnace gas alone is too diluted, and must be enriched with producer-gas to give the high temperature required for the reaction. When it is available, however, it is economical, as the coal-consumption in the gas-producer can thereby be cut almost in half. In the United States, crude oil is used at two plants and natural gas at two. Local conditions govern the choice of fuel.

For different fuels and for different ores the construction of the furnace must be somewhat modified, although broadly the design is governed by the same principles. In the first third of the furnace the briquettes are preheated by the escaping combustion-gases; they pass through zones of increasing heat as the cars are pushed forward. In the combustion-chamber the heat is maintained at about 2,500° F. When the briquettes have passed the combustion-chamber, they gradually pass through the cooling-chamber, into which air is blown by means of a fan; and when they leave the furnace, they are sufficiently cool to be loaded directly into railroad-cars. The arch over the cooling-chamber is double, and, in some installations, is made of corrugated cast-iron plates. Through the space between the double arches passes the air that is to be used directly for the combustion. On reaching the burners or the combustion-chamber it has a temperature of from 400° to 600° F. The outgoing combustion-gases generally reach the stack-flue with a temperature of from 350° to 500° F. It will be seen that the furnace is built on the regenerative principle, and that its heat-efficiency is good. The walls, as has been said, are double, and the arch is usually covered with 8 or 10 in. of sand, so that radiation-losses are low.

2. *Chemical Reactions.*

As a rule, regardless of their original composition, the briquettes are discharged as a peroxide of iron (Fe_2O_3). If the ingoing ore is a hematite, it is gradually transformed into Fe_3O_4, as the ore passes towards the combustion-chamber. In a nearly white-hot condition, it is here met by a current of highly-heated air. This gives the ideal conditions for a rapid conversion from Fe_3O_4 to Fe_2O_3. Here lies the secret of the Gröndal process. Briquettes at a temperature of 2,400° or 2,500° F. are met by air of high temperature. The oxidation is almost instantaneous,[6] and liberates an appreciable amount of heat, to the benefit of heat-conditions in the combustion-zone.

A quick oxidation of the Fe_3O_4 produces a blue, crystalline Fe_2O_3. A slow oxidation would give a reddish, earthy, amorphous product. Hence, it is necessary to maintain a high temperature in the combustion-zone. The color and condition of the discharged product tell whether the furnace is run properly. Dark blue, firm and strong briquettes indicate a right temperature; loose and reddish briquettes, a temperature too low.

When the ingoing material is a magnetite, it is probably oxidized to a certain degree on its way towards the combustion-zone. This oxidation is probably followed by a dissociation as the briquettes enter the hotter part of the furnace; but the Fe_3O_4 so formed is again oxidized to Fe_2O_3 before being cooled off and discharged. This last re-oxidation is so complete that analyses usually show more than 90 per cent. Fe_2O_3 of the total iron oxide in the briquette.

It is probably these chemical reactions which account for the agglomeration and the great strength of the briquettes. The material is not rolled together when in a plastic state, as in the nodulizing process; nor fused together, as in the blast-roasting, which gives a sintering-product usually of little strength; but in the re-oxidizing of the lower oxides of iron diminutive crystals of hematite are formed, growing out on the surface of each ore-grain in the briquette. These small crystals interlace with each other or grow together, forming a strong bond between the grains. If the ore is not crushed fine enough, or

[6] G. Gröndal, *Stahl und Eisen*, vol. xxxi., No. 14, p. 537 (Apr. 6, 1911).

consists of rounded pebbles, this intergrowing effect may not be obtained, and consequently the resulting briquettes will be loose and easily disintegrated. Such was the case in tests with the concentrated product from St. Lawrence magnetic sands. It was found necessary to pass a part of the iron sand through a grinding-mill in order to get some fine material to fill the voids between the larger pebbles. With pyrites-residues there has been the same experience. If strong briquettes are desired, it is futile to try to briquette cinder from lump-pyrites without first grinding it so as to pass, say, 20-mesh.

It has been mentioned that the briquettes in passing through the furnace are desulphurized. The conditions in the furnace are favorable to a complete desulphurization, namely: a high heat, a strongly oxidizing atmosphere, and a porous material consisting of finely-granulated ore-particles, permitting the gases to reach every diminutive grain. Moreover, the material stays in the hot zone for several hours, permitting a gradual oxidation of the sulphur, which escapes in the form of dioxide, with probably a small amount of trioxide. At Bayonne, cinder originally containing 2 or 3 per cent. of sulphur is discharged with about 0.03 per cent. Similar results are shown in the following table:

TABLE I.—*Chemical Results of the Gröndal Process.*

Works.	Crude Ore.			Concentrates.				Tailings.	Briquettes.		
	Iron.	Sulphur.	Phosphorus.	Iron.	Sulphur.	Phosphorus.	Recovery.	Iron.	Iron.	Sulphur.	Phosphorus.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Herrang, Sweden	40.2	1.21	0.003	67.3	0.170	0.002	92.9	6.4	65.5	0.003	0.002
Vigelsbo, Sweden	35.2	0.45	0.026	67.1	0.089	0.002	90.0	6.7	65.2	0.010	0.003
Strassa, Sweden	46.8	0.030	0.015	69.2	0.015	0.003	95.4	6.1	67.1	0.005	0.003
Lulea, Sweden [a]	58.2	0.110	1.230	71.1	0.015	0.005	95.5	12.0	69.3	0.005	0.005
Guldsmedshyttan, Sweden [a]	50.7	3.0	0.003	70.1	0.5	0.002	93.5	10.2	68.2	0.010	0.002
Flogberget, Sweden	27.3	0.31	0.003	67.4	0.040	0.003	82.7	7.1	65.3	0.007	0.003
Bredsjo, Sweden	35.0	0.15	0.010	67.2	0.050	0.004	89.5	6.9	65.1	0.020	0.004
Helsingborg (purple ore)	60.6	0.17	.....	.....	.....	.....	.....	.....	60.6	0.023	
Salangen, Norway	35.7	0.039	0.23	69.3	0.019	0.009	92.8	4.9			
Sydvaranger, Norway	38.0	0.066	0.030	69.0	0.026	0.006	92.8	5.5	67.0	0.006	0.006
Cwmavon, S. Wales (pyrites-residues)	64.2	2.79	0.019	.....	.....	.....	.....	.....	64.2	0.035	
Tharsis S. & C. Co., (Cardiff, England)	64.8	0.31	.....	.....	.....	.....	.....	.....	64.6	0.054	
Cornwall, Pa. [a]	39.5	2.5	0.031	68.9	0.51	0.011	72.0	19.2	66.9	0.010	0.014
Bayonne, N. J.	60.0	2.5	.....	.....	.....	.....	.....	.....	62.0	0.05	
Mayville, Wis.	46.2	0.035	.....	.....	.....	.....	.....	.....	56.0	0.01	
Moose Mountain, Canada	36.19	0.024	0.072	65.58	0.029	0.020	89.0	7.6	63.5	0.014	0.020

[a] Crude ore contains hematite, pyrites or other non-magnetic ferro-compounds.

3. *The Briquettes.*

I will mention here the results of only a few of the many tests made to determine the suitability of Gröndal briquettes for blast-furnace and open-hearth use.

Last year a low-grade magnetite was crushed and concentrated in a large commercial test. A part of the concentrate was briquetted at Bayonne, N. J. Of the product, about 7 tons were shipped in an open bottom-dump car to Youngstown, Ohio. Here, in unloading, the briquettes were dropped upon an iron floor from a trestle 30 ft. high. A screen-test of the unloaded material showed 8.84 per cent. of the total mass passed through a 0.5 in. screen. These fines gave the following results under a sieve-test:

	Per Cent.
On 8-mesh,	69.70
On 20-mesh,	16.66
On 40-mesh,	4.55
On 60-mesh,	0.81
On 80-mesh,	0.80
On 100-mesh,	0.60 + 0.06
Through 100-mesh,	6.66 + 0.16
Total,	99.78 + 0.22

This result shows that the total amount of fines passing through 40-mesh after the briquettes had been dropped 30 ft. upon an iron floor is only 0.8 per cent.

A regular compression-test of the same briquettes showed that they would withstand a pressure of 5,221 lb. per sq. in. That in handling they form a very small amount of dust was shown by the following test. A number of briquettes were reduced in a Gates crusher, to fragments 0.25 in. and less in diameter. A sieve-test of the crushed material gave only 10 per cent. through the 100-mesh sieve, and very little on the other fine sieves.

The porosity of the briquettes—that is, the ratio of voids to the total volume—is always more than 20 per cent. This is a strong point in their favor, since it aids in making them easily reducible in the blast-furnace.

4. *The Use of the Briquettes.*

John Jermain Porter, in his paper, The Fuel-Efficiency of the Iron Blast Furnace,[7] presented at the Wilkes-Barre meeting of the Institute, June, 1911, says:

"The great desirability of having an ore which is readily reduced by carbon monoxide rather than by solid carbon, and in addition is reduced at such low temperatures that the resulting carbon dioxide has no solvent power, has been frequently pointed out. The importance of carbon-deposition in this connection does not, however, seem to be so generally appreciated. It will be recalled that this reaction, $2\ CO = CO_2 + C$, begins at about 430° and ceases entirely at 900°. That is, it takes place very near the top of the furnace. It is probable that very little of the carbon resulting from this reaction ever reaches the hearth, but it does useful work in reducing the carbon dioxide of the limestone and in removing that portion of the oxygen of the ore which has not been removed by carbon monoxide higher in the furnace. From this point of view it appears that the ability of an ore to induce carbon-deposition is equally as important as the ease with which it loses its oxygen."

After recognizing the disadvantageous effects of an excessive deposition of carbon, Mr. Porter concludes by declaring that, these objections being "overcome by suitable design and management of the furnace, it is certainly true that every pound of carbon deposited means a saving of a pound of fuel for the hearth."

He also gives some numerical factors of reducibility for various classes of ores, to be used in formulas for finding the fuel-efficiency of the blast-furnace. These figures indicate that the Mesabi ores are most easily reduced, and these are followed, in the order of their reducibility, by brown hematites, soft red hematites and roasted carbonates, hard red hematites, Clinton "hard red" ore, and magnetites and mill-cinders, the last two being those which require most coke for their reduction.

The foregoing description of the physical and chemical character of the briquette produced by the Gröndal method, shows it to have all the qualities of an ore that can be reduced in the blast-furnace with the least coke. It is porous, so that by the dissociation of the carbon monoxide in the upper part of the stack, the fine carbon is deposited throughout its mass. Furthermore, the briquette is a hematite, and therefore not open to the usual (and, as can be seen from Mr. Porter's figures,

[7] *Trans.*, xlii., 191 (1911).

well-founded) objection of blast-furnace managers to the use of too large a percentage of magnetite in the burden.

It is the rapid reduction of the briquettes in the blast-furnace that has made them so popular in Sweden. The use of from 25 to 50 per cent. of briquettes in the burden of the charcoal blast-furnaces of that country has been shown to reduce by from 15 to 25 per cent. the charcoal-consumption per ton of pig-iron; 50 per cent. of briquettes in the burden giving the best charcoal-economy. In Sweden, from 500,000 to 600,000 tons of pig-iron are produced yearly. The annual product of briquettes, averaging 64 per cent. of iron, is from 300,000 to 350,000 tons, representing about 200,000 tons, or about one-third of the total production, of pig-iron. This means a yearly saving for the country of 400,000 cu. m. of charcoal,[8] an item of importance in these times of increasing scarcity of wood suitable for charring. If so great a saving has been made in charcoal-furnaces, it is not too much to believe that a corresponding saving in coke-furnaces is both practicable and important.

I have figures from a furnace in Belgium which has been using imported Swedish concentrate-briquettes in a mixture with calcined Bilbao spathic ore. With 70 per cent. of briquettes and 30 per cent. of Bilbao ore in the burden, the coke-consumption was 1,760 lb. per ton of pig-iron, whereas, with all Bilbao ore, it had been 2,398 lb. The iron-content of the Bilbao ore was 50 per cent.; of the briquettes, 65 per cent. By the use of the briquettes, the output of the furnace was increased from 20 to 25 per cent., with a corresponding saving in wages and general charges.

From the United States I have no definite figures. The briquettes produced here have been used in such small lots at various plants that no valuable data have been obtained. I have been recently informed, however, that at Mayville the coke-consumption in the blast-furnace is considerably reduced by their use. The briquetting-plant there has been in operation so short a time that it is too early to expect authoritative figures.

For use in open hearth-furnaces, the briquettes are emi-

[8] G. Gröndal, *Stahl und Eisen*, vol. xxxi., No. 14, p. 537 (Apr. 6, 1911).

nently fitted, being rich in iron, free from noxious substances, and in lump form, of sufficient weight to sink through the slag cover.

5. *The Practical Operation of the Gröndal Briquetting-Plant.*

The fine ore to be briquetted must contain sufficient moisture to maintain its brick-form after pressing. Too much moisture affects badly the work of the presses; too little makes the briquettes dusty and causes considerable spilling in pressing. The proper percentage of moisture varies with the hygroscopic qualities of the ore. Concentrated magnetites have to be de-watered to 8 or 9 per cent.; fine pyrites-cinder can carry from 15 to 18 per cent. and still give good results. The tempering is done in the simplest manner by a sprinkling-pipe or water-hose. The man who watches the feeding of the presses soon learns by the feeling of the ore whether it has the right moisture-content. The delivery of the ore to the presses is in modern plants performed mechanically at a small expense. In most of the present installations, drop-presses of the Dorstener type, developed by Dr. Gröndal, are used; but on account of the usually rather high cost of maintenance of these presses, it is likely that toggle-presses of heavy construction will be substituted in future plants.

The standard size of the briquettes is 6 by 6 by 2.5 in. Two tiers are loaded edgewise on each car, so that the load stands 12 in. high. At the Duquesne plant, briquettes of the size of standard red brick—2.5 by 4.5 by 8 in.—are made, and are loaded three, or even four, tiers high. Care is taken to place them on the cars so that the gases can penetrate the load and heat the bottom row as thoroughly as the top one. The loading is done by hand, and constitutes the only hand-labor around the plant. This work, however, is so laid out as to make it as easy as possible. The press-man removes the briquettes as they are delivered by the press, and places them on the car, which is pushed close to him. He does not need to move from his place during his work; and, although he handles only 10 or 12 lb. each time he turns, he can place 30 tons in a 12-hr. shift. The press delivers from 12 to 16 briquettes a minute, so that enough idle time is allowed for changing cars. The present endeavor is to eliminate this

manual labor, which, it must be admitted, is quite strenuous and, with wages at about $2.10 per day, represents from 7 to 10 cents of the briquetting-cost per ton.

The loaded cars are pushed through the furnace at intervals, one car being admitted at a time. This car pushes the others forward, so that, as the last car is introduced, the first is discharged from the furnace. The pushing, which is done by a hydraulic ram or some similar contrivance, requires little power. The discharged car is unloaded either by being pushed under a plow, which scrapes the briquettes over the sides of the car, or by being tipped endwise. Either arrangement can be made entirely automatic; so that, in a four-furnace plant, two men can take care of all the cars as they come out, unload them, and send them back to the presses over a return-track with endless-chain haulage. The frequency with which the cars are charged depends upon the nature of the ore. If it is high in sulphur, they may have to be charged on that account at a slower rate. At Mayville cars are charged every 10 min. The furnaces there are 195 ft. and the cars 6 ft. 6 in. long. Each furnace, therefore, contains simultaneously 30 cars, which require (since one is pushed in every 10 min.) 5 hours to pass through. At Bayonne the furnace is only 143 ft. long. One car is pushed in every 18 min., so that it takes 6.5 hours to pass through.

The length of the furnaces (which, as already shown, is quite different in the Mayville and the Bayonne plants) depends upon the nature of the ore to be briquetted. The width has been gradually increased. To begin with, it was not more than 3 ft. In the United States both 5-ft. and 6-ft. furnaces are used, and foundations have now been laid for two, having an inside width of 10 ft. Plans have been drawn for furnaces 15 ft. wide; and I see no reason to prevent even a greater width. A wide furnace, with a capacity of, say, 300 tons of briquettes per 24 hr., would show a still greater heat-economy than present plants. The development of the process is advancing in this direction.

Simultaneously, improvements in the design of the furnace are introduced in every new installation, in order to strengthen details which have proved defective in previous plants, and to reduce the cost of maintenance. Good results have been attained in these respects. The furnaces themselves will stand

for years without the walls or the arch having to be touched; and the repair-cost on cars amounts to only a few cents per ton. Mechanical labor-saving devices are introduced everywhere, so that the labor-cost is being lowered.

The fuel-consumption is low. In plants treating magnetic concentrates, the amount of coal used in the gas-producers averages 7 per cent. of the weight of the briquettes; and there are plants in Sweden using as little as 5 per cent. At a plant in the United States where pyrites-residues are briquetted, and crude oil is used as fuel, the oil-consumption is reported to be 15 gal. per ton of material briquetted. It is obvious that, if the ore is heavy, like magnetic concentrates, for instance, the percentage of fuel used, if calculated on the original charge, will be lower than in the treatment of ore that contains a large amount of combined water and other volatile matter. With such ore, a shrinkage in weight of from 15 to 20 per cent. has to be taken into consideration.

Electrostatic Concentration or Separation of Ores.

BY HENRY A. WENTWORTH, BOSTON, MASS.

(New York Meeting, February, 1912.)

ELECTROSTATIC separation of ores in its present form is generally known as the Huff process, from the name of Charles H. Huff, of Boston, Mass., through whose constant and persistent labors (with the invention of Clinton E. Dolbear as a basis) the successful commercial process embracing separative machinery and the various electrifying devices has been developed step by step, and the finances for the long period of development provided, and the method finally established and recognized throughout the world as an important and successful addition to the ore-dressing department of metallurgy.

The permanent field-success of electrostatic separation began in 1908 with a 20-ton Huff experimental mill built specially for the purpose by the American Zinc, Lead & Smelting Co., in Platteville, Wis., a plant which was a success from the start and was gradually increased in capacity as the market-conditions warranted to 100 tons of concentrates per day. Much credit is due to the above-mentioned company for its initial

venture, and for its assistance in applying the process to field use.

Prior to 1908 electrostatic separators had been installed and operated (but for a comparatively short time, however) in a number of places; some under the patents of Mr. Dolbear by himself and associates, and some under the patents of Lucien I. Blake and Lawrence N. Morscher, by W. G. Swart, mining engineer, of Denver, who has always been a courageous advocate of electrostatic separation.

Due to the difficulties experienced in the generation of the electrical charges, the primitiveness of the separators, the wooden construction instead of iron as at present, the lack of control of the electrical fields and other difficulties overcome by the later inventions of the Huff Electrostatic Separator Co., electrostatic separation did not gain favor during those early field-endeavors.

There is an old experiment in physics where an electrified rod is brought close to a suspended pith ball. The pith ball is first attracted, clings for a moment to the rod, and is then vigorously repelled. As the rubber rod approaches the pith ball, a charge of opposite kind, so called, is induced on the side of the pith ball nearest to the charged rod, and as unlike charges of electricity attract one another and as the pith ball is very light, it moves to the rubber rod. But pith, though not a good conductor of electricity, does, because of the moisture contained, conduct electricity appreciably, and it soon becomes, as a whole, charged similarly to the rubber rod, and away it flies. This is the principle which is utilized in electrostatic separation, and to accomplish separation, the differential property is the conductivity of the minerals involved. Thus, if the circuit of an ordinary 110-volt incandescent lamp is broken, and a piece of pyrite inserted in the break, the current will again flow. Similarly, if a piece of quartz be used, the electrical circuit will remain broken. Between these extremes, the conductivities of minerals grade, there being, however, a great difference in conductivity between the so-called better conductors and poorer conductors.

The physical separation may be accomplished in several ways, all utilizing the same underlying principle of difference in conductivity of the minerals. In the simplest form, a me-

chanical mixture consisting of readily-conducting and poorly-conducting minerals (such, for example, as grains of copper and grains of sand) is dropped on to a metallic plate charged to a high potential. Immediately upon contact with the plate the better-conducting minerals become charged to the potential of the plate, and are thrown vigorously from it. The poorer conductors require a much greater time to reach the electrical condition of the plate, and therefore, if they are not given time to reach this condition, due to being removed from the plate, a separation is obtained. This method was utilized in the simplest form of electrostatic separator, as illustrated in Fig. 1, in which the charged plate is replaced by an electrified roll.

Wherever there is electrical repulsion, there must be an electrostatic field (analagous to a magnetic field), with two terminals to the lines of force. Just as a magnetic particle endeavors to move along the magnetic lines of force, and would were there no other forces acting, so an electrified particle endeavors to move along the electrostatic lines from one terminal of them to the other. In the example just cited, the charged surface is at one terminal of the lines, while the walls of the room are at the other, the electrostatic lines existing as is shown in Fig. 2.

The particle charged from its contact with the metallic roll, may be said to be repelled from the roll, or attracted to the walls of the room.

Suppose now instead of permitting all of the lines to pass to the walls of the room, they are concentrated to another roll, as shown in Fig. 3, by connecting opposite terminals of the charging-device to the two rolls. The field is now much more localized and intense than in the case of the disseminated field of Fig. 2.

However, instead of depending solely upon the removal of the poorly-conducting particles from the surface of the charging-body before they have time to become charged, the length of time during which the metal surface remains charged may be regulated, as the better-conducting particles obtain their charges practically instantaneously upon contact.

Still another method of applying and utilizing the charge may be used. It has long been known that from points or sharp edges of conductors, when charged to a high potential,

there emanates an electrical spray, which consists of charged air passing from the point or edge to the surface of opposite potential. Assume now there are passing over the roll a number of grains of good conductors and poor conductors,

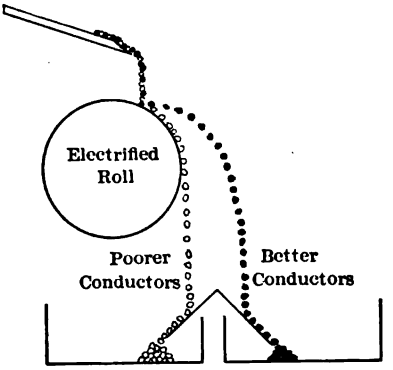

FIG. 1.—SIMPLEST FORM OF ELECTROSTATIC SEPARATOR.

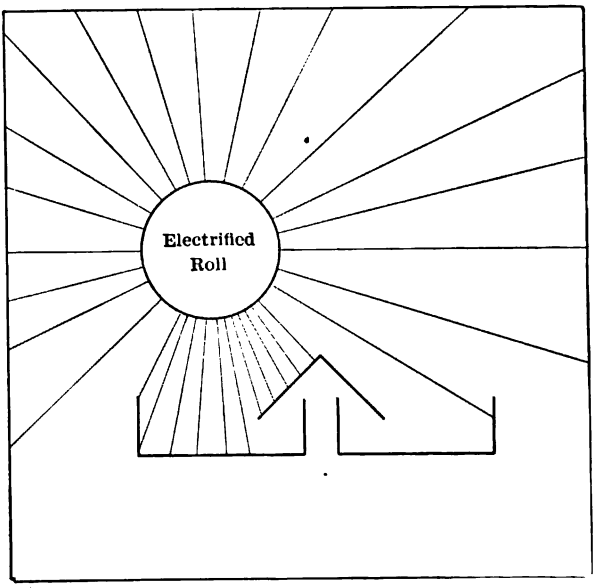

FIG. 2.—LINES OF ELECTROSTATIC FORCE.

as illustrated in Fig. 4. The charged air traveling from the point to the roll will strike upon the backs of these particles which are in contact with the roll and deliver charges to the particles. In the case of good-conducting particles this makes

no difference, as the charge is immediately transferred from each particle to the roll with which it is in contact; but with the poorly-conducting particles the charge leaks away but slowly to the roll, with the result that the charges on the roll and on the backs of the particles are different, and the particles are held firmly to the roll until the charge leaks away. Assume the roll is grounded. The charges on the backs of the poorly-conducting particles induce charges on the face of the roll near them, and the endeavor of the two kinds of charge to unite holds the particles closely and tightly to the

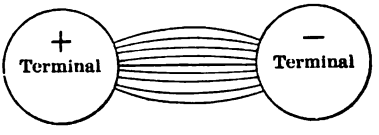

FIG. 3.—CONCENTRATION OF ELECTROSTATIC FIELD.

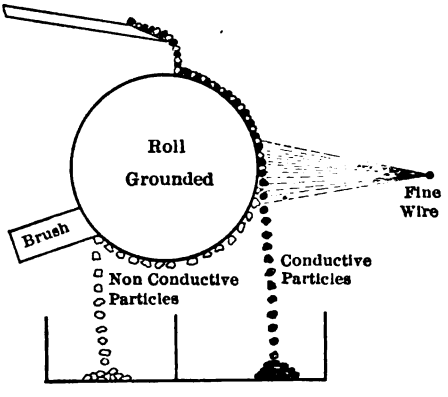

FIG. 4.—CHARGES DELIVERED BY ELECTRIC SPRAY.

roll. When the roll has made a partial revolution the particles are removed by a brush, while the differently conducting minerals may be separated by the use of gravity or centrifugal force.

For years these various methods of application of the principle have been studied, and a process developed to combine the applications in the manner which seemed to give the best results in the field under the conditions of mill-practice. It should be borne carefully in mind that in all the cases above

cited the separative effect is accomplished by taking advantage of the difference in electrical conductivity of the particles.

Several types of Huff separators have been developed for application to different materials. The following description is of two types only—those in most common use and most generally applicable. A type consisting of six separating-fields and a feeder is illustrated in Fig. 5, which does not, however, show the large feed-hopper and the heavy base. The entire machine, with the exception of the attracting-rods and their supports, is made of metal, and is electrically grounded, so that all portions

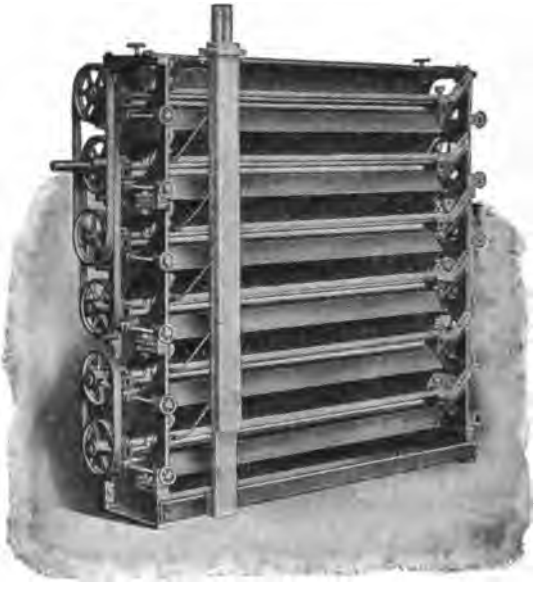

FIG. 5.—HUFF ELECTROSTATIC SEPARATOR WITH SIX SEPARATING-FIELDS.

which come in contact with the ore are metal, and all parts with which the operator is likely to come in contact are grounded so that the operator gets no electric shocks in touching the machines. As it is desired to pass a thick sheet of ore through the separators, there is more or less interference of particles at each separating-roll; hence it has been found advisable to use several rolls in succession, each contributing a part towards the complete separation. This is especially necessary in those cases in which it is desired to keep the ore coarse at the beginning, and thus have many attached particles to be

reground. In order to have all the metallics come in contact with the separating-surface, a number of contacts at different angles are required. In this machine, the only moving parts are seven steel rolls revolving in babbitted and greased bearings, so that the power necessary to drive the separator itself is extremely small.

Fig. 6 is a diagrammatic sketch of a second form of separator, which is finding excellent favor in the separation of washed zinc-iron middlings below 20-mesh, which flow very readily. The advantage of this type is that there are no moving parts whatever, except the feed-roll, and therefore there is a minimum of wear and required power. Not all material, how-

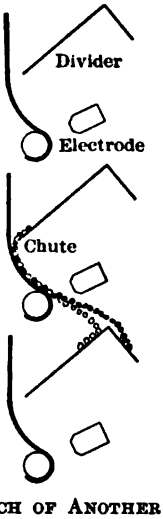

FIG. 6.—DIAGRAMMATIC SKETCH OF ANOTHER FORM OF HUFF SEPARATOR.

ever, will flow sufficiently freely without external assistance to be applicable to this type of separator.

The size of particles which can be successfully treated by the present forms of Huff machines is from 6- or 8-mesh down to the limit of granularity of the material. Those slimes which are so cohesive that they do not move uniformly over inclined chutes cannot as yet be successfully handled. It is possible that this difficulty can be overcome by a special design of feeder or separator. In Utah a table-middlings is being treated wherefrom the impalpably fine material has been removed during wet-table treatment. However, of that portion going to the "C" (finest size) machines, all of which is through 80-

mesh (aperture 0.0082 in.), 20 per cent. passes a 240-mesh (aperture 0.0020 in.) screen. Huff machines have been built to handle material much coarser than 6-mesh, but up to the present the demand for such a machine has not warranted the investigation necessary to develop it completely.

The action produced in the usual type of electrostatic separator consists in electrically giving the relatively conductive particles a horizontal component of motion in addition to the motion produced by gravity. The less conductive particles, not being thus affected, are acted upon by gravity alone. The heavier the conductive particle, the stronger must this repellent force act to cause it to fall on the outer side of the divider of the machine. The repellent force is dependent upon the intensity of the electrostatic field. As electrostatic separation treats particles varying greatly in size, therefore, with a field sufficiently strong to repel the very heavy particles, there is danger of the finer less conductive particles being thrown over also. Hence it is advisable not to have a feed in which the particles are too widely diverse in size, and therefore in weight. In practice the feed is screened into a few sizes, of which screening the following is an example: from 8- to 12-; from 12- to 20-; from 20- to 50-, and through 50-mesh. When the difference in conductivity of the minerals is small, it is sometimes advisable to size somewhat more closely.

At this point attention is called to a matter which properly belongs later. Crushing some varieties of ore to 10-mesh, for example, will expose practically all the mineral, but will free but little of it. Because of the numerous separation-fields to which the material is subjected while passing through the separator, all sides of every particle are brought into contact with the charging-surface, and nearly every particle which contains an appreciable portion of exposed mineral will be repelled, thus eliminating the rock as coarsely as possible. All the mineral can be thrown into a concentrate, or the better part into the concentrate from the first two or three rolls, and the balance into a middlings for recrushing.

The question of the electrification of the separators in a manner which should not be affected by varying atmospheric conditions was a serious problem at the beginning of the electrostatic development. The frictional or induction generators

are in themselves exceedingly susceptible to changes of humidity of the atmosphere, and also their capacity is so small that any source of electrical loss on the line or in the separators is likely to throw the whole system out of adjustment. It is unfortunate that this difficulty should have been so widely advertised in the earlier stages of the work in connection with the "Blake" machines. It has restrained many from investigating the Huff machines. There is no longer any difficulty whatever from this source, the machines being adaptable, if properly installed, to any climate or any atmospheric conditions. The electricity is furnished the Huff machines by electromagnetic generators instead of frictional machines. These generators work as independently of atmospheric conditions as does the ordinary lighting-dynamo, and are capable of supplying any line- or machine-losses which may occur and yet hold the potential absolutely the same from day to day. Contrary to general belief, the difference of potential which is used in electrostatic separation is only from 18,000 to 25,000 volts, instead of about 100,000 volts, earlier supposed to be necessary. There has been developed a small, compact, 3-h-p. electrical set, which is sufficiently large to care for the requirements of a 100-ton mill, and probably a much larger plant. Protective devices are placed between the generators and the separators to prevent injury to workmen in case of accident at the separators or accidental contact with the high-potential line. In the four years of its field-operation, no one has been injured from this source.

As the effectiveness of any electrostatic separation depends on the differences in conductivity of the particles, it follows that there must be no extraneous factor interfering to affect the conductivity. Water is, in the sense of electrostatic separation, a good conductor. Therefore the particles must be dry. Some minerals dry very readily and remain so. Some are hygroscopic, and if allowed to stand cold for some time in a moist atmosphere, tend to collect on their surfaces an extremely thin film of moisture. However, experience has shown that by properly constructing the mill equipment so that the ore from the drier passes immediately through its various paths, and is not permitted to lie for some time cold, and exposed to the atmosphere of a room, very few minerals, and none of the common ones, offer any serious difficulty from this standpoint.

There seems to be a very erroneous idea of the cost of drying ore. Crude mine-ore will run from 1 to 5 per cent. of moisture; 3 per cent. being a fair average. This means 60 lb. of water to be evaporated from each ton of ore. A fairly-efficient drier will expel from ore carrying 3 per cent. of moisture, 5 lb. of water per pound of coal, or, say, 12 lb. of coal per ton of ore. With coal at $5 per ton, this means a fuel-cost of $0.03 per ton. Fine middlings from water-tables will sometimes carry as high as 15 per cent. of moisture, and the fuel-cost is therefore increased under those circumstances to 10 or 11 cents per ton with the $5 fuel-cost, as with the higher per cent. of moisture the evaporating-power goes up to 8 lb. of water per pound of coal.

With regard to the minerals and their conductivities, as stated previously, the condition is entirely relative, some minerals being better conductors than others. However, the minerals may be divided quite closely into two classes: one, those which are very easily charged and repelled, and the other, the very poor conductors which act as non-conductors. A complete list of the conducting minerals has been compiled by G. W. Pickard. Table I. presents the more common of both classes.

TABLE I.—*Conducting and Non-Conducting Minerals.*

IMPORTANT CONDUCTIVE MINERALS.

Argentite,	Galena,	Psilomelane,
Arsenic, Native,	Graphite,	Pyrargyrite,
Arsenopyrite,	Hausmannite,	Pyrite,
Bismuth, Native,	Hematite,	Pyrolusite,
Bismuthinite,	Ilmenite,	Pyrrhotite,
Bornite,	Jamesonite,	Redruthite,
Brookite,	Leucopyrite,	Silicon,
Calaverite,	Linnaeite,	Smaltite,
Carborundum, (artificial)	Magnetite,	Sperrylite,
Cholcopyrite,	Manganite,	Stannite,
Chalcocite,	Marcasite,	Stephanite,
Cobaltite,	Mercury, Native,	Sylvanite,
Copper, Native,	Millerite,	Tellurium,
Covellite,	Molybdenite,	Tetrahedrite,
Enargite,	Niccolite,	Wad,
Ferrosilicon,	Pentlandite,	Wolframite,
Franklinite,	Proustite,	Zincite.

POORLY-CONDUCTIVE MINERALS.

Zinc-Blende,	Epidote,	Apatite,
Quartz,	Garnet,	Rutile,
Feldspar.		

Nearly all the silicates, carbonates, and sulphates.

Most of the siliceous rocks.

There are a few important minerals whose conductivity is variable, being dependent upon the composition of the particular specimen. Among these are notably blende and garnet. Pure sphalerite is a very poor conductor, ranking among the best insulators. It seldom occurs pure, however, but is usually contaminated chemically by varying amounts of iron sulphide or manganese sulphide. When these impurities are present in very large quantity in the blende, the resulting mineral is commonly called marmatite. The behavior of blende in electrostatic separation is somewhat dependent upon the amounts and character of impurities in the mineral. Small amounts of iron or manganese, and sometimes up to several per cent. of these impurities, have no seriously deleterious effect upon the behavior of the blende, but when the impurity is present in sufficient amount the resultant mineral changes over into the class of conductors. The relation of impurity and conductivity seems to follow no definite law, but each sample must be examined independently. From the above, however, it is readily seen that it is theoretically impossible to produce the same grade of finished blende-product in any manner from all samples of ore. Whereas the Joplin or Wisconsin blende crystals will analyze from 66 to 67 per cent. of zinc, those of some regions when mechanically free from all impurity contain only from 40 to 45 per cent. of zinc, and perhaps from 10 to 20 per cent. of iron. As a rule, electrostatic separation is very successful in separating the zinc-minerals from other minerals.

Similarly, the conductivity of garnet is sometimes dependent upon the amount of iron present. A high-iron garnet is likely to be more conductive than a low-iron garnet. A study of the data in Table I. will show the general field for electrostatic separation or concentration.

Considering a few specific problems and the specific reasons for the peculiar adaptability of the Huff process to these problems, it should be noted, as shown in Table I., that most of the copper sulphide minerals are excellent conductors, while most gangue-rocks are poor conductors. Therefore, electrostatic separation is applicable to the general concentration of the sulphide ores of copper. It is particularly well-adapted to the concentration of copper-minerals from the heavier gangue-min-

erals, such as garnet, epidote, barite, the heavy basic rocks, etc. Large bodies of ores of this character exist, usually as altered contact-deposits, though, except in cases where the grade is sufficiently high for direct smelting, they have been developed but comparatively little, because difficulty has heretofore been found in concentrating them.

Often a combination of methods is more efficient than a single process, the governing factor being whether the size of operations warrants complexity of procedure. Very large copper-mills employing gravity-concentration produce a concentrate which has been carried to the economical limit of purification by the method employed, but which is yet capable of mechanical improvement if a different property of the ore than its specific gravity be worked on. Electrostatic separation offers a means of reducing the silica and other objectionable constituents of the gravity-mill concentrates.

Zinc-blende usually occurs in mechanical association with pyrite or marcasite, chalcopyrite, galena, and one or more gangue-materials. The specific gravities of these minerals are approximately as follows: galena, 7.5; pyrite, 5.0; marcasite, 4.8; chalcopyrite, 4.2; blende, 4.; ordinary gangues, 2.7. From the smelters' standpoint, it is essential (for maximum recovery of the metal at minimum cost) that the material going into the lead-smelter be as free from zinc as possible; that into the copper-smelter, free from zinc and lead; and that into the zinc-smelter, as high in percentage of zinc as possible. Of the latter, theoretically, 67 per cent. is the maximum, but only in exceptional cases (as in the Joplin and Wisconsin districts) is 60-per cent. zinc-blende product obtained commercially by any method, while 50-per cent. zinc is for most complex ores considered excellent and 45-per cent. good. With ores in which the various mineral ingredients dissociate at a reasonable (30-mesh or coarser) degree of crushing, gravity-separators (jigs or water-tables) will usually effect a reasonably efficient separation of minerals which differ in specific gravity by 1.5 points, but as the difference in specific gravity becomes smaller the effectiveness of separation in this manner becomes less, so that with a difference of but one point, the effectiveness of the separation is poor. By putting such a complex ore, suitably crushed, through jigs and over reciprocating tables, the

galena (with a portion of the pyrite or marcasite) is efficiently concentrated from the rest of the minerals. Likewise, the elimination of the gangue-rocks is also reasonably well effected. There are left together a greater portion of the pyrite or marcasite, the blende, and the chalcopyrite (with of course a little gangue and a small amount of galena). Fortunately, the galena (which is usually so small in amount as to be negligible except for the associated silver which often occurs in it), the pyrite or marcasite, and the chalcopyrite are excellent conductors of electricity, while the blende and the small amount of gangue-rock are very much poorer conductors. In the utilization of this difference of conductivity is seen the application of the electrostatic process to this problem. This mixture, after drying (but not roasting), is passed through the Huff machines, and two products made, one for the zinc-smelter and one for the copper-smelter.

Blende often occurs in association with certain heavy gangue-rocks of a specific gravity so similar to that of the blende that separation by any gravity means is very ineffective. A method has been developed by the Huff company whereby the surface of the blende can be made conducting, after which the blende can be separated electrostatically from the associated rock-minerals. Two important instances of the above are blende and barite, and blende and fluorspar.

In addition to the concentration or separation of the sulphides mentioned heretofore, the Huff process has proved that it also accomplishes the following results very satisfactorily: concentration of gold and silver pyritic ores, of antimony, arsenic, and molybdenum sulphides; concentration of graphite, of pyrite for sulphuric acid manufacture, hematite, manganese-ores; the separation of galena and barite; purification of abrasives, natural and artificial; and the solution of many problems relating to the more rare minerals. Copper oxides, carbonates, and silicates can sometimes be concentrated, by first roasting to the conductive oxide, or reducing to the metal.

The process is essentially a dry one, obviating the troubles due to drought or freezing. The finished products require no further drying-treatment prior to shipment or smelting to decrease freight-expenses or fuel-costs. Loss in slimes is avoided, as the dust may be collected and is available for use if desired.

There are no shaking or vibrating parts to the machines with the attendant wear and repair on the mechanism. The attendance necessary, after primary adjustment, is small and ordinary mill-labor is suitable. The machines are readily sectionalized for light transportation and easily assembled. There are no complicated or intricate parts to get out of order and the separators are readily adjustable while operating.

The Huff electrostatic process was started into field-operation in the spring of 1908, at the beginning of the present general slump in the mining business. Nevertheless, the method has made progress, and wherever installed has been very successful. The first plant mentioned has been already described.[1]

At Midvale, Utah, the United States Smelting Co. has a very complex lead-zinc-copper-gold-silver ore, for which has been installed an electrostatic plant producing 50 tons daily of zinc- and iron-concentrates, the procedure in use being that above described for complex ores containing zinc. This plant also is described in the references quoted.

A further instance of the eminent adaptability of the Huff process to zinc-ore problems is the newly-constructed electrostatic mill at the Sunnyside mine, Silverton, San Juan county, Colo. This is treating about 40 tons per day, improving the zinc-concentrates obtained in the gravity-milling of the ores of the district.

There is also a 40-ton plant recently installed in Sonora, Mexico, which is giving excellent satisfaction in the separation of blende-chalcopyrite concentrates, producing a blende-product containing 55 per cent. of zinc and a copper-product assaying about 16 per cent. of copper.[2]

A plant recently installed by the Carborundum Co. at Niagara Falls, N. Y., is separating an impurity from artificial abrasive.

The cost of construction and of operation depends upon the general costs in the given region, upon the size of operations,

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[1] *Transactions of the American Electrochemical Society*, vol. xviii., p. 267 (1910); *Mining World*, vol. xxxiii., No. 23, p. 1041 (Dec. 3, 1910); *Engineering and Mining Journal*, vol. xc., No. 1, p. 15 (July 2, 1910); *Metallurgical and Chemical Engineering*, vol. viii., No. 11, p. 636 (Nov., 1910).

[2] *Engineering and Mining Journal*, vol. xcii., No. 23, p. 1080 (Dec. 2, 1911).

upon whether the plant is run as an independent concentrator or as an adjunct to other operations. Therefore total-cost figures are of but little value unless presented with a complete description of the conditions of operation. The following detailed items are presented, which, by addition to suit the circumstances, will give a proper total.

Only the concentration- or separation-department is here considered, as the grinding-department is the same, independent of the method of concentration. A flow-sheet for the treatment

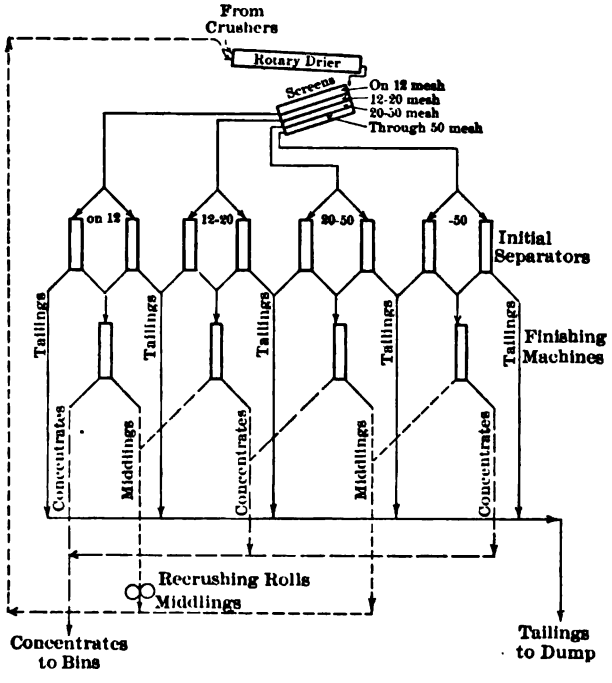

FIG. 7.—FLOW-SHEET OF HUFF ELECTROSTATIC-SEPARATION PROCESS.

of an ordinary crude ore in the concentrating-department, shown in Fig. 7, is as follows: rotary drier (not roasting), screens, roughing-machines to make a finished tailings; finishing-machines making finished concentrates, and a small middlings for return to the general system. The dust is drawn from each piece of apparatus and collected (to be used if sufficiently rich, and discarded if of little value). As the separators are upright, they occupy a space 6 by 1.5 ft., and are about 7 ft. high. Leaving sufficient space for attendance, a machine

occupies a space 8 by 5 ft., or 40 sq. ft. of floor-space per separator. From 9 to 12 tons per separator can be taken as an average on the initial machine-floor, so that the tonnage-area of the building is about 4 sq. ft. per ton for the separators and about 6 sq. ft. per ton including elevators, belts, etc. The Midvale plant, with a daily capacity of 75 tons of very fine material, occupies a building but 40 by 40 ft. in area, but there is a very considerable amount of space unused at present. The drier is usually situated in the basement, and the screens in a tower above the concentrator-floors. For a crude mill of 1,000 tons daily capacity, approximately 6,000 sq. ft. of building is necessary, or about 60 by 100 ft. The other costs of installation are similar to those of the ordinary gravity-concentration, except that the elevators do not have to handle water. Also, because of the small size of the building, there is a minimum of distributing-machinery.

The operating-costs outside of crushing-costs consist of: drying, power, labor, and repairs. The drying has been mentioned already, as has also the electrical power necessary for separation. The mechanical power for driving the separators is figured at $\frac{1}{3}$ h-p. per separator, including line-losses, so that in large installations the entire power, including elevators, screens, etc., amounts to approximately $\frac{1}{6}$ h-p. per ton of daily capacity. As everything in a properly-designed mill is automatically handled, with the exception of firing the drier, the labor required is needed only for properly keeping watch of the mill and for loading the products.

In general, the costs of operation in an electrostatic mill do not differ materially from those of a similar mill using reciprocating tables.

The Huff Electrostatic Separator Co. maintains at its Boston office a department for the investigation of various ore-dressing and allied problems.

The James Diagonal-Plane Slimer.

BY S. ARTHUR KROM, PLAINFIELD, N. J.

(New York Meeting, February, 1912.)

THE James diagonal-plane slimer is specially adapted to handle the finest slimes, but it will also handle sands as coarse as 40-mesh. The saving efficiency of this machine is due to the original construction of the deck, and to the rapid reciprocating movement imparted to it by the head-motion.

The deck consists of a plurality of connected planes, having a certain fixed ratio of inclination one to another. These planes intersect each other diagonally to the stroke-line of the table and are arranged so as to give the pulp on the table a double treatment. This double concentration performed by the diagonal-plane deck produces cleaner concentrates and tailings than those obtained by the operation of two separate machines. The action of the diagonal planes settles, combines, and holds all the concentrates in one compact mass, sharply defined from the gangue, and easily cut away without loss.

Many slime-concentrating machines distribute the feed-pulp for a distance of 4 or 5 ft. only, and upon a very limited settling-surface. This results in a thick, heavy deposit, from which it is impossible to stratify the values and wash away the gangue, the resulting products being dirty concentrates, and tailings carrying high values.

The diagonal-plane slimer distributes the feed-pulp for a distance of 10 ft., and upon a settling-surface having an area of 40 sq. ft.; in other words, it spreads the pulp thinly on the table, which allows the metallic values to settle through the gangue quickly, forming a bottom stratum from which the thinly-distributed overlying stratum of gangue is easily brushed away by the feed-water without disturbing the underlying concentrates.

Referring to Fig. 1, plane *A* is the main settling-section of the table. The feed pulp flows gently down this plane and is

not retarded until it reaches the lower edge of the plane, at which point the settled minerals are collected by the plane *B*, having less inclination than plane *A*, the feed-water and gangue flowing over them. A small portion of the gangue settles with the minerals. The main portion of the gangue is checked by planes *C*, called the "retarding-planes," and forms a buffer which assists the motion to form the minerals in a narrow, thick bed, and to drive them along the line of intersection of planes *A* and *B* towards their discharge-point. Upon reaching the retarding-planes *C*, the settled values are moved faster by the table-motion towards their discharge-point than the downward flow of the feed-water is moving them towards the tailings stratum, making the tendency to escape from their own formation very slight.

Planes *C* terminate 4 ft. short of the discharge-end of the table. At this point plane *B* gradually increases in width to the discharge-end of the table, forming a cleaning-area on which the minerals spread in a thin stratum, allowing the wash-water to carry away the remaining gangue. This gangue flows over the greater inclined plane *D*, together with the middlings and gangue discharged from the retarding-planes *C*, and the whole is arrested by the plane *E* and the retarding-planes *F*, and treated in the same manner as the pulp caught by the upper planes. The values settled and caught by the planes *E* and *F* are driven by the table-motion into, and consolidated with, the lower edge of the main concentrates stratum formed by the upper planes. With the proper amount of wash-water and feed, it is very difficult for the values to escape from all the planes.

The head-motion, illustrated by Fig. 2, is constructed to relieve the rapidly-rotating parts from the working-strains as much as possible. These strains are reduced by a system of levers from 6 to 1. In other words, if the table was bolted directly to the rotating-mechanism, these parts would have six times more pressure on them than under the lever construction.

Index-plates are provided as a guide to change the length and character of the stroke for various sizes of coarse and fine pulp.

A stroke-card of the table is shown in Fig. 3, in which curve 1 represents the stroke as set for treating fine minerals, the

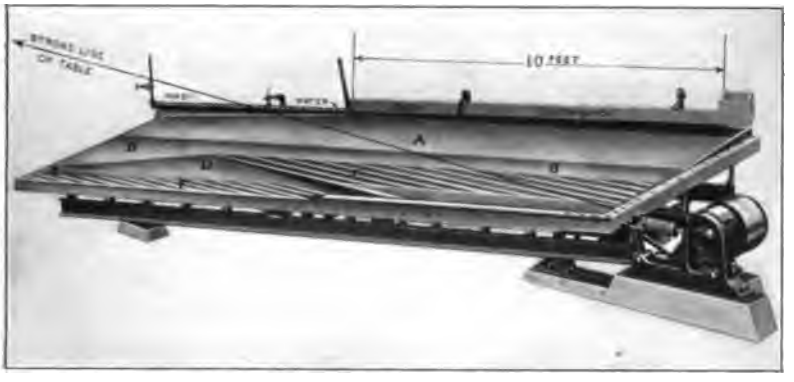

FIG. 1.—DECK-VIEW OF JAMES DIAGONAL SLIMER.

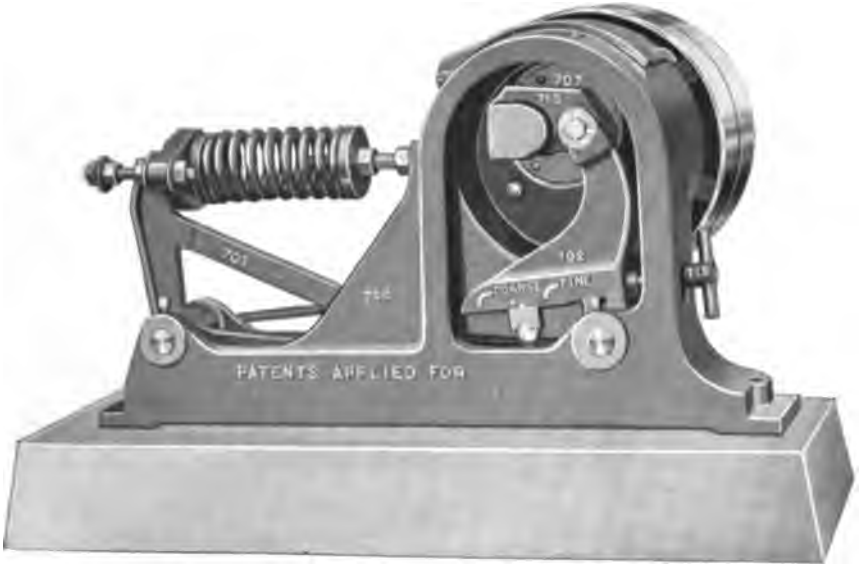

FIG. 2.—HEAD-MOTION OF JAMES DIAGONAL SLIMER.

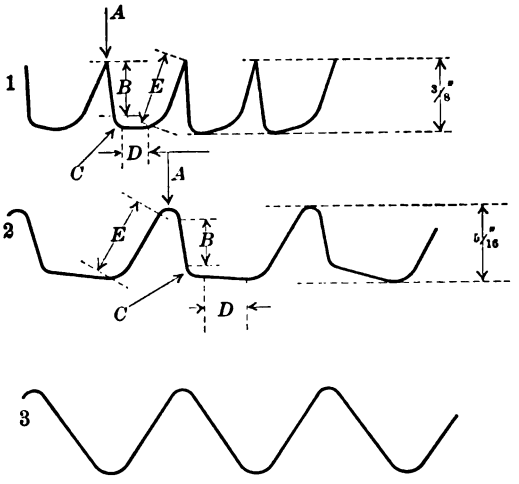

FIG. 3.—TABLE-STROKE CARDS.

FIG. 4.—VIEW SHOWING METHOD OF SUPPORTING THE JAMES SLIMER DECK BY THE USE OF 42 CARRIERS.

length of stroke being about $\frac{3}{8}$ in. The sharp point of the outline at *A* shows the quick reversal of the table for the back stroke. The perpendicular part of the line at *B* indicates the rapidity of the back stroke to the curve *C*. During the passage of the stroke at *C* it slows down, coming to almost a dead stop while passing over *D*. This quiet action at *C* and *D* avoids throwing the fine pulp into such violent agitation that it would not have time to settle before the beginning of the forward stroke. The settlement being completed during the slow passage over *D*, the forward stroke begins, and gradually increases in speed as shown by curve *E*, until point *A* is reached again.

The stroke as set for coarse pulp is shown in Fig. 3, curve 2. It will be noticed that the reversal of the table for the back stroke at *A* is not so sharp as in the case of the slime-stroke; this slow reversal prevents packing and jamming the coarse pulp particles together, which would occur with a sharp reversal on this material. The quick portion of the reverse stroke at *B* is shorter, and the passage over *D* is faster, than in the treatment of slimes, owing to the fact that the settlement of the coarse pulp is not so easily disturbed and less time is required for settlement, hence the forward stroke can be started sooner than when treating slimes. Also, the speed of the forward stroke, it will be seen, is slower, in order to prevent driving the table from beneath the coarse pulp and failing to advance it.

When working fine material the stroke-reversal at *A* is 12 times faster than its speed at *D*, while in the case of the coarse pulp it is only four times that at *D*.

The revolutions per minute of the table being constant under all conditions, the length of the stroke must be kept within certain limits, otherwise the table-surface speed would increase to a point producing agitation sufficient to throw the pulp clear of the table, and prevent its settlement during the passage of the stroke over *D*. Thus the table would fail to advance the material properly through lack of contact.

Curve 3 illustrates the stroke produced by the regulation table toggle movement, the construction of which only allows a change in the length of the stroke. With this exception, coarse and fine pulps receive the same treatment.

Variations in the character of the stroke, as shown by curves 1 and 2, make it possible to produce cleaner products. The

capacity of the table is also increased, due to the fact that the separations are made quicker.

It has been found in practice that by changing the character of the stroke, separations can be made, assisted by the diagonal planes, of minerals heretofore very imperfectly concentrated in table-practice. Such separations as zinc from fluorspar, graphite from gangue, etc., are made very clean for commercial purposes by this combination.

The following is a comparison of results obtained by vanners and tables with riffled deck and toggle motion, and by the diagonal plane and differential motion, working the same feeds, in mill-practice.

	Tailings.
Riffles and toggles, ...	0.42 per cent. copper
Diagonal planes, ...	0.26 per cent. copper
Riffles and toggles, extraction (lead), . .	60 per cent.
Diagonal planes, extraction (lead), . .	80 per cent.

	Extraction.	Concentrates.	Tailings.
Frue vanner, .	50 per cent.	50.0 per cent. lead	5 per cent. lead
Diagonal planes,	64 per cent.	48.8 per cent. lead	3 per cent. lead

Frue vanner, extraction (cobalt-silver), . .	40 to 50 per cent.
Diagonal planes, extraction (cobalt-silver), .	76 to 82 per cent.

	Lead-Concentrates. Contain	Zinc-Concentrates. Contain
Riffles and toggles, .	10 to 15 per cent. zinc	9.0 per cent. lead
Diagonal planes, .	9 per cent. zinc	0.92 per cent. lead

Fig. 4 shows the method of supporting the table-deck. Forty-two carriers are hinged to the table, a canvas belt forming the hinge. The same material attaches the carriers to the channel-irons of the foundation frame. This construction shows no signs of wear after five years, running 24 hr. per day.

The construction of the deck itself is unusually light, and has a double advantage over the regulation heavy deck-construction, namely, the power required to oscillate the table is greatly reduced, and the deck is easily drawn to its proper bearings when being installed, eliminating all warping that may have occurred during transportation.

The Rational Valuation and Quality-Efficiency of Furnace-Stock.

BY JOHN JERMAIN PORTER,* CINCINNATI, OHIO.

(New York Meeting, February, 1912.)

THE value of any particular ore, coke, or limestone, for iron-making, depends upon its effect, first, upon the quality or value of the resultant product; and second, upon the cost of smelting. The factors under the first head are the percentages of phosphorus, manganese, and other elements, which remain in the pig-iron; while those under the second head are the percentages of iron, carbon, or lime, the slag-forming constituents, which must be fluxed, and any other ingredients which give rise to variations in cost of fuel and flux.

The first of these criteria of value is perhaps not capable of being reduced to a general formula embracing all cases, since it must vary to a great extent with market-fluctuations. For example, Bessemer pig is worth sometimes more and sometimes less than foundry-pig; and the premium paid for high manganese varies within wide limits. The second, or smelting-value, however, can be expressed numerically with a fair degree of accuracy; and it is with this phase of valuation that the present paper is concerned.

The utility of a means of comparing numerically the relative smelting-values of different ores, cokes, and limestones, is self-evident, although it may not have importance in Northern practice, where materials are largely standardized. On the other hand, there are, especially in the South, many furnace-managers who must use a great variety of materials of widely varying grade; and it is with special reference to this district, and in connection with the efficiency-methods which I have elsewhere proposed,[1] that the methods herein described were devised.

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[1] Cost Reduction in the Iron Industry, *Manufacturers' Record*, vol. lix., No. 5 (Feb. 9, 1911).

Previous work along this line has been confined chiefly to the valuation of ores. Moreover, none of the methods which I have examined[2] have met the requirements of my own particular needs. The methods which follow lack somewhat in scientific completeness, and depend to a regrettable extent upon factors which must be chosen empirically by experience. Nevertheless, I claim for them the two requisites of practical usefulness, namely, simplicity and substantial accuracy.

Method for Iron-Ores.

The basis of this method is the percentage of iron, silica plus alumina, lime plus magnesia, and the ease of reduction as measured by the burden-ratio which can be carried in average practice.

Let Fe = percentage of iron.
RO = percentage of lime plus magnesia.
V = value of iron per unit.
v = value of coke per short ton.
q = value of RO per unit.
Ac = percentage of silica plus alumina.
n = percentage of Ac in coke ÷ 100.
m = long tons of coke per ton of ore.

$$r = \text{burden-ratio} = \frac{\text{ore} + \text{stone}}{\text{coke}} \qquad (1)$$

The term "unit" refers to 1 per cent. of a long ton, or 22.4 lb.

Value of ore per ton = value of Fe + value of RO — value of RO to flux Ac — value of coke to smelt. (2)

[2] The Valuation of Iron-Ores, by M. Drees. *Stahl und Eisen*, vol. xxvii., No. 10, p. 330 (Mar. 6, 1907).

The Valuation of Ores, by A. Rzehulka. *Zeitschrift für angewandte Chemie*, vol. xxiii., No. 11, p. 481 (Mar. 18, 1910).

Observations on the Purchase of Ore, by C. Chordot. *Revue de Métallurgie*, vol. viii., No. 3, p. 289 (Mar., 1911).

Comparison of Some Southern Cokes and Iron Ores, by A. S. McCreath and E. V. D'Invillers. *Trans.*, xv, 734 (1886–87).

Notes on the Selection of Iron-Ores, Limestones and Fuels for the Blast-Furnace, by F. W. Gordon. *Trans.*, xxi., 61 (1892–93).

Rules for Adjusting Premiums and Penalties on Lake Superior Iron Ores, by Geo. Smart. *Iron Trade Review*, vol. xlviii., No. 10, p. 495 (Mar. 9, 1911).

The Blast Furnace and the Manufacture of Pig Iron, by Robert Forsythe. 1st ed., pp. 47, 85 and 89 (1908).

The Calculation of the Value of the Raw Material in Pig Iron Making, by A. P. Gaines. *The Iron Age*, vol. lxxiii., No. 15, p. 12 (Apr. 14, 1904).

From (1), long tons of coke to smelt 1 ton of ore =

$$\frac{1 + \text{stone per ton of ore}}{r} \tag{3}$$

Assume that in the slag Ac = RO. This is so nearly true as to introduce only an inappreciable error, even with widely varying practice. Also assume that two tons of stone give one ton of RO.

Then tons of stone to flux coke-ash = 2 nm;
and tons of stone per ton of ore =

$$2\left(\frac{Ac}{100} - \frac{RO}{100}\right) + 2\ nm. \tag{4}$$

Hence, substituting (4) in (3),
Long tons of coke per ton of ore =

$$\frac{1 + 2\left(\frac{Ac}{100} - \frac{RO}{100}\right) + 2\ nm}{r} \tag{5}$$

and short tons of coke per ton of ore =

$$\frac{1 + 2\left(\frac{Ac}{100} - \frac{RO}{100}\right) + 2\ nm}{1.12\ r} \tag{6}$$

Hence, substituting in (2),
Value of ore = V Fe + q RO — q Ac — v

$$\left(\frac{1 + 2\left(\frac{Ac}{100} - \frac{RO}{100}\right) + 2\ nm}{1.12\ r}\right) \tag{7}$$

This method gives the true smelting-value of an ore, except that it does not take into account differences in the desirability of the slag made, due to the ratio of lime to magnesia, or of silica to alumina, or to the presence of titanium. If desired, account can be taken of the exact relation between the per cent. of bases and acids in the slag, and of the exact number of tons of limestone to give one ton of RO. However, these refinements make the formula considerably more cumbersome to use, and are unnecessary for substantial accuracy.

It will be noted that n is the percentage of acids (silica and alumina) in the coke, not the per cent. of ash. A considerable proportion of coke-ash is usually iron, which does not need fluxing. Occasionally there is a large amount of lime and magnesia in the ash; and in such cases it would be proper to consider n as the difference between the acids and the bases, or in other words, as the unfluxed acids. However, this is a matter of minor importance. Such changes as are likely to occur in q, n, and m have but little effect on the results, the two important factors being the value given to the unit of iron and the price of coke. Since the value of the iron-unit varies somewhat in different iron-making districts and at different times, it is my custom, when starting to compare a series of ores in this way, to try several values of V on some typical ore, finally adopting that value which gives results approximating actual prices. The following values for the various constants generally give good results:

V = 0.07 to 0.09.	v = $2 to $3.
q = 0.01.	n = 0.10.
r = 1.5 to 2.5.	m = 0.6.

The following is a comparison by this method of typical ores used in leading manufacturing centers of the world. In order to compare them, I have assumed that they are smelted under the same conditions and have used the following constants: V = $0.08, v = $3, q = $0.01, n = 0.10, m = 0.6. In one sense this assumption is misleading, since commercial value is very largely a function of cost of fuel and market for product. Hence an ore may be worth considerably more than indicated here, if located adjacent to cheap coke and a good market. This objection does not, of course, apply to the use of the method in comparing ores for any given furnace or district.

Ore.	r.	Fe.	Ac.	RO.	Value.
Lake Superior, Mesabi, . . .	2.5	53.0	11.0	——	$2.70
Alabama, hard red ore, . . .	1.7	35.0	15.0	18.0	1.17
Brown hematite, Virginia, . .	2.3	45.0	20.0	——	1.64
Magnetic concentrates, . . .	1.5	60.0	12.0	1.0	2.32
"Ironstone," Cleveland, Eng., .	2.1	32.0	15.0	11.0	1.02
"Minette" ore, Germany, . .	2.3	35.0	15.0	15.0	1.51

METHOD FOR COKE.

The value of coke for use in the blast-furnace depends on:

1. The percentage of carbon, since it is only the carbon which by its combustion produces heat.

2. The ease with which the coke is dissolved by carbon dioxide, since this determines in part the amount of carbon reaching the hearth of the furnace.

3. The percentage of ash, since the ash not only displaces carbon, but requires lime to flux it, and additional carbon to melt the slag formed from it.

The percentages of sulphur and of phosphorus may be disregarded in this connection, as not affecting the heating-value of the coke, although, of course, they affect the quality of the iron made from it. It may be objected here that the presence of much sulphur necessitates the use of a more basic slag and a higher temperature in the hearth, and in this way increases the amount of necessary fuel. Yet, while the truth of this statement must be admitted, its consideration in a workable formula does not seem practicable, and it is not often a matter of great importance.

A general expression for the value of coke is:

Value per short ton = value of carbon reaching hearth — value of carbon to smelt slag from ash — value of RO to flux ash.

Let FC = percentage of fixed carbon in coke.

Ac = percentage of silica + alumina in coke.

f = factor of solubility of coke in carbon dioxide.

r′ = reducibility-factor of ore.

RCO3 = pounds of lime and magnesia carbonates per ton of iron.

F = fuel-consumption in short tons of coke per ton of iron.

Ha = heat available in hearth of furnace per pound of carbon burned.

U = value of each unit of available carbon.

q = value of each unit of RO (long-ton units).

The term unit refers to 1 per cent. of one short ton, or 20 lb., except where otherwise noted.

The units of carbon reaching the hearth = FC — carbon-loss between throat and hearth.

The carbon-loss between throat and hearth I have shown elsewhere[3] to equal, per ton of iron, 0.12 × pounds of carbonates per ton of iron × size-factor of stone × factor of solubility of coke + 700 × reducibility-factor of ore × factor of solubility of coke. The size-factor of stone is unimportant and may be neglected.

Hence, units of carbon lost per short ton of coke =

$$(0.12 \times \mathrm{RCO3} \times \mathrm{f}) + (700 \times \mathrm{r}' \times \mathrm{f}) \times \frac{1}{20\,\mathrm{F}}$$

or, simplifying $= \frac{\mathrm{f}}{20\,\mathrm{F}}(0.12\ \mathrm{RCO3} + 700\ \mathrm{r}')$.

Hence, value of carbon reaching hearth =

$$\mathrm{U}\left(\mathrm{FC} - \frac{\mathrm{f}}{20\,\mathrm{F}}(0.12\ \mathrm{RCO3} + 700\ \mathrm{r}')\right) \text{ per short ton.}$$

The slag resulting from the ash is 2 Ac units per ton of coke. The carbon to smelt one unit of slag is (from the paper previously cited) $\frac{600}{\mathrm{Ha}}$.

Hence the value of the carbon to melt the slag from the coke-ash

$$= \mathrm{U}\left(\frac{2\ \mathrm{Ac}}{1} \times \frac{600}{\mathrm{Ha}}\right) = \mathrm{U}\left(\frac{1200\ \mathrm{Ac}}{\mathrm{Ha}}\right)$$

Finally, the short tons of RO required to flux the coke-ash = Ac (if we assume, as in the case of ores, that in the slag the per cent. of Ac equals the per cent. of RO), or the long ton units of RO = 1.12 Ac, and the value of the RO = q × 1.12 Ac.

Hence the value of the coke per short ton =

$$\mathrm{U}\left(\mathrm{FC} - \frac{\mathrm{f}}{20\,\mathrm{F}}(0.12\,\mathrm{RCO3} + 700\,\mathrm{r}') - \frac{1200\ \mathrm{Ac}}{\mathrm{Ha}}\right) - 1.12\ \mathrm{q\ Ac.}$$

Values of constants which I have found to give good results are as follows:

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[3] The Fuel-Efficiency of the Iron Blast-Furnace, *Trans.*, xlii., 191 to 221 (1911).

$U = 0.04$ to 0.06.
$q = 0.01$.
$f = 0.5$ for the best Connellsville coke.
0.6 to 0.7 for good Pennsylvania and West Virginia coke.
0.8 to 0.9 for fair Virginia and Alabama coke.
1.0 for the softest Pocahontas and Alabama coke.
$r' = 0.1$ for Mesabi ore.
0.2 for brown hematites.
0.3 for soft red hematites and roasted carbonates.
0.4 for hard red hematites.
0.6 for Alabama hard red ore.
1.0 for magnetites and mill-cinders.

The factor f simply represents that proportion which the carbon actually dissolved bears to that which could be possibly dissolved by the CO_2 present. For example, if there are 700 lb. of carbon present as CO_2 per ton of iron, then according to the reaction $C + CO_2 = 2\ CO$, 700 lb. is the maximum amount of carbon which can be dissolved per ton of iron. If good Connellsville coke with a solubility-factor of 0.5 is used, the amount of carbon actually dissolved and thereby wasted will be $700 \times 0.5 = 350$ lb., while with fair Alabama coke the amount lost would be $700 \times 0.8 = 560$ lb. The values given for this factor are based on no experimental evidence (except in a qualitative way), but were chosen because they give good results in furnace-calculations. I believe they are a fair approximation to the truth.

The factor r′ represents that proportion of the oxygen of the ore which is only removed at such high temperatures that the resulting CO_2 has power to dissolve carbon. It requires approximately 700 lb. of carbon to reduce the iron in one ton of pig-iron by the reaction $Fe_2O_3 + 3\ CO = Fe_2 + 3\ CO_2$. If this CO_2 were produced at such a high temperature that it all reacted thus: $CO_2 + C = 2\ CO$, there would be 700 lb. of carbon dissolved in this way, and r′ is the proportion of this maximum loss which actually occurs. The values used are in qualitative accordance with laboratory-experiments, and give good results when used in furnace-calculations; otherwise they have no scientific basis.

It will be noted that another kind of reducibility-factor is used here than in the case of iron-ores. It must be admitted that this is a very unscientific procedure; but the use of the burden-ratio to express relative reducibility finds its justification in the much greater simplicity of the calculations in the case of iron-ores. I have not as yet worked out the scientific relationship between r and r′, but an approximate empirical relation is given by the expression $r' = \frac{5}{r^4}$.

Ha, the heat available in the hearth of the furnace per pound of carbon, has been discussed by Johnson[4] and by me.[5] Its calculation is very lengthy, but by means of the diagrams in the two papers just referred to it can be read off directly. Ha is usually about 1,500 for Alabama practice, 1,600 for Virginia practice, and 1,400 for Northern practice.

Method for Limestones.

Let q′ = value of lime and magnesium carbonates per unit of 22.4 lb.

v = value of coke in dollars per short ton.

Ac = per cent. of silica plus alumina in the stone.

Ha = heat available in the hearth of the furnace per pound of carbon.

FC = per cent. of fixed carbon in the coke.

Relative value = value of available carbonates — value of coke to flux slag from siliceous impurities.

Available carbonates = 100 per cent. — per cent. Ac — per cent. carbonates to flux Ac.

Per cent. carbonates to flux Ac (assuming RO = Ac in slag) = 1.785 Ac if pure calcium carbonate, or = 1.911 if pure dolomite. Say = 1.8 Ac for average practice.

Hence available carbonates = 100 — Ac — 1.8 Ac. = 100 — 2.8 Ac.

Short tons of coke to melt slag per ton of stone =

$$\frac{\text{pounds of slag per ton stone} \times 600}{\text{Ha} \times \frac{\text{per cent. FC}}{100} \times 2000}$$

Pounds of slag per ton of stone $= \frac{2\ \text{Ac}}{100} \times 2240.$

[4] *Trans.* xxxvi., 476 (1905).

[5] *Trans.*, xlii., 193 (1911).

Therefore the short tons of coke to smelt the slag per ton of stone =

$$\frac{\frac{2\ \text{Ac}}{100} \times 2240 \times 600}{\text{Ha} \times \frac{\text{per cent. FC}}{100} \times 2000},$$

$$\text{or simplifying,} = \frac{1344\ \text{Ac}}{\text{Ha} \times \text{per cent. FC}}.$$

Therefore value of stone =

$$q' \times (100 - 2.8\ \text{Ac}) - v \times \frac{1344\ \text{Ac}}{\text{Ha} \times \text{per cent. FC}}.$$

Examples of the Use of these Formulæ.

During a portion of the year 1909 a certain Southern furnace used materials of which the following are the extremes in composition, as shown by the best and poorest analyses on the furnace-books.

		Iron. Per Cent.	Silica + Alumina. Per Cent.	Lime + Magnesia. Per Cent.
Ore 1, hard red ore,	best,	46.54	6.90	11.54
	poorest,	33.00	23.40	12.50
Ore 2, hard red ore,	best,	36.30	19.40	12.25
	poorest,	33.03	26.92	11.29
Ore 3, soft red ore,	best,	50.70	19.10	2.40
	poorest,	38.60	34.80	2.00
Ore 4, soft red ore,	best,	49.20	27.60	.....
	poorest,	35.77	45.00	.....
Ore 5, brown ore,	best,	44.10	21.00	.....
	poorest,	39.29	28.30	.....
Ore 6, brown ore,	best,	41.10	26.05	.....
	poorest,	30.10	46.30	.....
Ore 7, brown ore,	best,	49.60	18.20	.....
	poorest,	36.04	34.02	.....
Limestone 1, . . .	best,		3.01	
	poorest,		9.57	

		Fixed Carbon. Per Cent.	Ash. Per Cent.	Silica + Alumina. (Approximate). Per Cent.
Coke 1, ...	best,	85.80	13.75	10.75
	poorest,	75.65	22.25	19.25
Coke 2, ...	best,	82.22	16.58	13.58
	poorest,	77.80	20.58	17.58

The various constants entering into the calculation of the relative values of these materials may be taken as follows:

$V = 0.09$	$v = \$3.00$
$q = 0.01$	$n = 0.15$ (average.)
$m = 0.6$	$U = 0.05$
$r = 1.7$ for hard red ore.	$F = 1.7$
$r = 2.0$ for soft red ore.	$Ha = 1500$
$r = 2.3$ for brown ores.	$f = 0.9$
$FC = 80$ per cent. (average.)	$r' = 0.5$ (average of all ore used.)
$q' = 0.008$	$RCO_3 = 2600$

Hence the equations for this particular furnace and period may be simplified to the following:

Value of ore =

$$0.09\ Fe + 0.01\ (RO - Ac) - \frac{3.9 + 6\left(\frac{Ac - RO}{100}\right)}{1.12\ r}$$

Value of coke = $0.05\ (FC - 17.5 - 0.8\ Ac) - 0.0112\ Ac$.
Value of limestone = $0.012\ (100 - 2.8\ Ac) - 0.0336\ Ac$.

The relative values of these materials, figured on this basis, together with the approximate actual costs delivered at the furnace, are as follows:

	Relative Value. Best Material.	Relative Value. Poorest Material.	Approximate Actual Cost.	Relative Desirability (*i. e.* Value ÷ Cost). Best Material. Per Cent.	Relative Desirability (*i. e.* Value ÷ Cost). Poorest Material. Per Cent.
Ore 1, . . .	$2.33	$0.50	$1.20	194.2	41.7
Ore 2, . . .	0.93	0.27	1.20	77.5	22.5
Ore 3, . . .	2.17	0.49	2.00	108.5	24.5
Ore 4, . . .	1.63	0.23	1.50	108.6	15.3
Ore 5, . . .	1.78	1.10	2.00	89.0	55.0
Ore 6, . . .	1.34	0.32	1.75	76.6	18.3
Ore 7, , . .	2.36	0.61	2.00	118.0	30.5
Limestone 1,	0.63	0.27	0.60	105.0	45.0
Coke 1, . .	2.86	1.92	3.00	95.3	64.0
Coke 2, . .	2.54	2.11	3.00	83.7	70.4

As a matter of fact in this particular case, the ore-situation was such that almost anything was welcome that would help keep the furnace going; and the operating officials were fully aware of the inefficiency of some of the material used. Never-

theless, I think that the value of such a comparison as the above will be generally admitted.

One of the great difficulties in the way of economical production is the frequent lack of co-ordination between different departments. It is quite commonly the case that foremen have as their ideals the good of their own departments rather than the good of the whole organization; and it is indeed an efficient executive who can hold all the details of a large operation so firmly as to eliminate all waste due to friction between the parts.

In the South especially, a considerable amount of unnecessary waste is often due to wrong ideals on the part of the miners and mine-foremen. Finding that success is gauged almost entirely by the elements of cost and output, it is natural that they should devote all their energies to making a good showing along these lines, while quality is neglected, except perhaps spasmodically when the blast-furnace superintendent makes some particularly strenuous objection. The latter is usually the only person who is particularly interested in this question of quality; and he is seldom in a position to determine what standards are proper and how much of the poor material used was really unnecessary.

It appears to me that, in such cases, efficiency-reports, if properly constructed, should be of special value in giving a true basis upon which to evaluate the work of each department, and in helping each member of the organization to more useful ideals. My idea for carrying out this plan, as worked out in connection with other phases of furnace-efficiency, involves the use of three reports. The first, Table I., is a periodic, preferably daily, blast-furnace stock-efficiency report, made up from the analyses of the materials as shipped, and preferably sampled at the furnace. The second, not shown, is the daily operation-report of each mine, washer and coking-plant, and would have to vary in its make-up according to the nature of the operation. The third, Table II., is a weekly or monthly comparative mine-efficiency report. It will be noted that on this the efficiency as to cost, output, and quality is shown separately, while the total efficiency is the product of these three. Of course there are many possible variations in carrying out this idea, and every case will probably need a

little different treatment. Bonus-payments for high efficiency to those in charge are a very desirable, though not absolutely necessary, corollary to the plan.

It is not within the province of the present paper to consider the question of standards for cost and output; but the matter of quality-standards can, in the case of coal, limestone, and bedded ores, be handled as follows: At regular periods, say once a week, a careful and reliable man goes through the mine and takes an adequate sample representing all, or a majority, of the working-faces. This sample excludes all partings which should theoretically be thrown out in mining, and its analysis, representing the best quality possible if the mining is perfectly done, becomes the standard for that week and affords the basis for calculating the standard for relative value. In the cases of brown-ore deposits, ore- and coal-washers, and coke-ovens, special methods of sampling must be devised for each individual case.

It may be objected that this plan calls for an undue amount of analytical work. This, however, need not be the case if the work is carefully planned; and, in any event, the benefits which are usually to be derived will warrant a considerable addition to the laboratory expense. As an example, I may cite the case of a company which has been using coke averaging 18 per cent. in ash, although it was demonstrated, when special attention was given to this feature, that their coal is capable of making a coke of only from 13 to 14 per cent. of ash. This increased ash is responsible, in that case, for an increase of approximately 300 lb. of fuel per ton of iron and an increased cost of about 40 cents per ton of iron.

On the other hand, I am acquainted with a coal-operation where the adoption of methods somewhat similar to, though less complete than, those I have advocated, has led to a decrease in the average ash in the coal shipped from 16 to 12.60 per cent. within a period of three months. Finally, it is well known that a leading Southern producer has been able to attain most remarkable results both as to low ash and uniformity of composition in its coke by the use of methods which contain the spirit, if not the outward form, of those I have here described.

TABLE I.—*Daily Stock-Efficiency Report.*

DAILY STOCK EFFICIENCY REPORT.

.....Furnace. Date,.....

Ore.		Iron.	Insoluble.	Bases.	Relative Value.	Efficiency.
		Per Cent.	Per Cent.	Per Cent.		Per Cent.
No. 1.	Standard.	46.54	6.90	11.54	2.53	
	Actual.	33 00	23.40	12.50	0.50	21.4
No. 2.	Standard.	36.30	19.40	12.25	0.93	
	Actual.	33.03	26.92	11.29	0.27	29.0
No. 5.	Standard.	44.10	21 00	.....	1.78	
	Actual.	39.29	28.30	.....	1.10	61.8
Flux.						
No. 1.	Standard.		3.01		0.63	
	Actual.		9.57		0.27	42.8

Coke.		Ash.	Fixed Carbon.	Relative Value.	Efficiency.
		Per Cent.	Per Cent.		Per Cent.
No. 1.	Standard.....	13.75	85.80	2.86	
	Actual.....	22.25	75.65	1.92	67.1
No. 2.	Standard.....	16.58	82.22	2.54	
	Actual.....	20.58	77.80	2.11	83.1

NOTE.—It is contemplated that the relation between the percentage of silica plus alumina and the per cent. of insoluble, and the per cent. of silica plus alumina in the ash, be determined and occasionally checked, and that a suitable instruction-sheet be furnished whereby the chemist can calculate the relative value with the minimum of labor.

TABLE II.—*Weekly Comparative Mine-Operation Report.*

WEEKLY COMPARATIVE MINE-OPERATION REPORT.

Week ending.....

Operation.		Cost.		Output.		Quality.		Total Efficiency.
			Efficiency.	Tons.	Efficiency.	Relative Value.	Efficiency.	
		Dollars.	Per Cent.		Per Cent.		Per Cent.	
No. 1.	Standard.	0.80		300		1.78		
	Actual.	0 90	88.9	310	103.3	1.10	61.8	56.7
No. 2.	Standard.							
	Actual.							
No. 3.	Standard.							
	Actual.							

The Refining of Blister-Copper.

BY HORACE H. EMRICH, KYSHTIM, RUSSIA.

(New York Meeting, February, 1912.)

PREFATORY NOTE.—The first part of this paper was sent to me by Mr. Emrich nearly nine months ago; and I held it, waiting for the second part, which he had promised to forward soon, so that I might submit the whole to some of his former professional associates, and advise him of their criticisms and suggestions before offering his work to the Institute. But, five days before the second part reached me, I received by telegraph the news of Mr. Emrich's sudden death. The paper has therefore not had the benefit of such a revision by the author as might have made it in some respects clearer and more complete. Nevertheless, I believe that experts in the branch of metallurgy with which it deals will find it both original and suggestive.—A. EILERS.

I. PARTIAL REFINING IN THE ANODE-FURNACE.

During my connection with one of the large New Jersey copper-refineries, and probably before that period, from 15 to 20 per cent. of the copper-bullion received was the very foul, leady blister-copper produced from lead blast-furnace matte. It had been customary to mix a small part of this blister with each charge of purer pig-copper, in the effort to dilute it as much as possible before casting into anodes; but tank-house conditions were such that I was led to try making entire furnace-charges of this blister, and refining it as much as possible in the furnace—afterwards casting to anodes and refining electrolytically, in tanks and solution separate from the main bulk of the purer copper.

In making this experiment, notes were made, for several furnace-charges, of everything thought to have a bearing on the question, and some interesting facts were brought out, both in the furnace-working and afterwards in the tanks.

The changes taking place during the furnace-refining are shown by the results of working two typical charges. The refining was done in a magnesia-lined, reverberatory refining-furnace of the usual design, built to cast 150 tons per charge.

Each lot of copper charged was taken at the settlement-weight, as were also the assays and analyses; each slag pulled off was weighed, broken to fist-size to remove all copper shot, coned and quartered; and a barrelful (a clean oak barrel was used) was sent to the mill for final grinding and cutting down to the sample. All copper shot and metallics regained were weighed and added to the total of copper regained, since only the analysis of the slag itself was wanted. The sample, representing the copper as finally cast into anodes, was a composite of two plates cast from the pouring-ladle at different times during the tapping of the furnace. All copper not accounted for was reckoned as remaining in the furnace, with the same analysis as the anodes.

The various steps taken during the working of the copper and the results are as follows:

Charge A.

Tapped June 14, 1909. Copper Charged into Furnace.

	Cu.		Pb.		As.		Sb.		Total Weight.
	Per Cent.	Lb.	Per Cent.	Lb.	Per Cent.	Lb.	Per Cent.	Lb.	Lb.
Lot No. 1.....	93.8	22,872	0.40	98	0.35	85	1.0	244	24,384
Lot No. 2.....	93.8	80,491	0.78	672	1.41	1,213	0.42	362	86,090
Lot No. 3.....	93.87	74,616	0.37	294	1.67	1,329	0.35	278	79,489
Lot No. 5.....	91.18	81,172	0.64	552	1.61	1,388	0.35	302	86,189
Lot No. 6.....	94.14	82,822	0.96	844	1.16	1,020	0.31	273	87,978
Total weight....		341,976		2,460		5,035		1,459	364,130
Average assay..	93.91		0.676		1.38		0.40		

The method of procedure in refining was thoroughly to oxidize the copper and attempt to remove lead, arsenic, and antimony by a copper-oxide slag. The pig was oxidized during melting by blowing compressed air, at 15 lb. pressure, through a 0.75-in. iron pipe thrust below the molten copper. With the large amount of lead present, the oxidation took place rapidly.

Each slag, as skimmed, was weighed and sampled, with the following results:

	Cu.		Pb.		As.		Sb.		Total Weight.
	Per Cent.	Lb.	Per Cent.	Lb.	Per Cent.	Lb.	Per Cent.	Lb.	Lb.
First slag.....	44.2	2,187	8.9	441	2.3	114	3.3	164	4,958
Second slag..	38.7	2,950	10.2	778	2.3	175	3.5	267	7,632
Third slag....	36.6	1,600	9.8	428	2.3	101	3.3	145	4,374
					Total slag.....				16,954

Average Assay of Slag Skimmed.

Au.	Ag.	Cu.		Pb.		As.		Sb.	
Oz. p. t.	Oz. p. t.	Per Cent.	Lb.	Per Cent.	Lb.	Per Cent.	Lb.	Per Cent.	Lb.
0.107	84.2	39.7	6,737	9.7	1,647	2.3	390	3.4	576

Removed from the furnace in slag: 1.97 per cent. of the Cu; 67.1 per cent. of the Pb; 7.7 per cent. of the As; 39.6 per cent. of the Sb.

It will be noticed that the amount of Sb driven off is very large; undoubtedly there is an error in the determination of Sb in the slag. It is probable that the results are double what they should be. (See analysis of slag drawn from charge B.) But this annoying error as to Sb does not affect the main argument, which is concerned chiefly with As. From the above figures it is seen that if the rate of the slagging of copper be taken as unity, the lead, arsenic, and antimony went into the slag at the following faster rates:

	Lead.	Arsenic.	Antimony.
First slag, . .	28.0	3.5	17.5?
Second slag, . .	36.6	4.0	21.2?
Third slag, . .	37.3	4.3	21.3?

Obviously, more slagging could have been done to advantage.

After cleaning off all slag, the copper was covered with coke and poled. It proved impossible to overpole this copper, though, in an endeavor to do so, on one charge (run later) 40 poles were burned, while the surface of the metal was kept well covered. Undoubtedly this was due to the action of the remaining lead as an oxygen-carrier.

Throughout the run, the surface of the copper gave off much smoke; and in casting so much was given off that the tapper complained of its making him feel somewhat sick. Even with a clear fire and when casting, a large amount of white smoke went off through the stack; of course, there was a great deal more during refining-operations. The figures given below show this to have been undoubtedly lead and arsenic, probably as oxides.

The total tapped from the furnace, including ladle-drip, slag, metallics, etc., was:

Au. Oz. p. t.	Ag. Oz. p. t.	Cu. Per Cent.	Cu. Lb.	Pb. Per Cent.	Pb. Lb.	As. Per Cent.	As. Lb.	Sb. Per Cent.	Sb. Lb.	Total Weight. Lb.
263	304.5	96.36	331,045	0.15	515	1.15	3,951	0.33	1,134	344,551
Copper unaccounted for, ...				.....		.....		.....		4,194
Equivalent to anode-copper left in furnace at 96.36 fineness, . .				.....		.....		.....		4,352
Containing, ...					16	.....	50	.....	15	
Total impurities accounted for, .					2,168	.....	4,391	.....	1,724	

From these figures it is seen that the antimony-analysis is incorrect somewhere (probably being reported in the slag as twice what it should be), since they show more antimony accounted for than was in the original pig.

During all the poling-operations, some copper is driven off through the stack; and, as there is no ready means of determining how much this is for any charge, all difference has been assumed to have been left in the furnace, and, on the basis of the assay of the anode, the impurities in this residual copper are calculated as accounted for. On this assumption, the difference between the total impurities accounted for and the amount charged, must have been driven off in the fume while melting and refining, as follows: Pb, 11.85 per cent.; As, 12.70 per cent.; Sb, (?), of the total in the pig charged.

There was removed, therefore, in all the furnace-refining operations, 1.97 per cent. of the copper, 78.9 per cent. of the lead, 20.6 per cent. of the arsenic, and ? per cent. of the antimony in the pig originally charged; and, of the total impurities removed, 15 per cent. of the Pb, 62.2 per cent. of the As, and ? per cent. of the Sb, went off in the fume.

For the purpose of comparison the results of a second similar charge are given below. It will be seen that practically the same things occur.

Charge B.

Copper Charged into the Furnace.

	Cu. Per Cent.	Cu. Lb.	Pb. Per Cent.	Pb. Lb.	As. Per Cent.	As. Lb.	Sb. Per Cent.	Sb. Lb.	Total Weight. Lb.
Lot No. 12....	93.54	45,746	0.61	298	1.06	518	0.41	200	48,905
Lot No. 13....	93.13	41,316	1.51	670	1.26	559	0.36	160	44,364
Lot No. 14....	93.31	72,887	1.49	1,163	1.58	1,235	0.38	297	78,113
Lot No. 15....	92.95	23,075	0.63	156	2.14	532	0.64	159	24,825
Lot No. 17....	93.34	76,488	0.99	811	1.13	926	0.42	344	81,946
Lot No. 18....	92.58	60,599	1.68	1,100	1.32	864	0.43	281	65,456
Lot No. 19....	92.05	19,010	2.04	421	1.66	343	0.46	95	20,652
Total weight....		339,121		4,619		4,977		1,526	364,261
Average assay..	93.1		1.27		1.37		0.42		

This charge appears to carry the same amount of arsenic per ton as charge A.

The slag skimmed was as follows:

Au. Oz. p. t.	Ag. Oz. p. t.	Cu. Per Cent.	Cu. Lb.	Pb. Per Cent.	Pb. Lb.	As. Per Cent.	As. Lb.	Sb. Per Cent.	Sb. Lb.	Total Weight. Lb.
				First Slag.						
0.06	25.8	34.7	1,711	15.8	779	2.8	138	2.3	113	4,930
				Second Slag.						
0.14	38.0	34.3	2,127	16.5	1,024	2.6	161	2.0	124	6,202
				Third Slag.						
0.18	36.8	34.8	1,141	16.1	528	2.3	75	2.0	65	3,279
				Fourth Slag.						
0.07	17.5	15.4	297	20.1	388	0.3	6	0.1	2	1,930
				Total slag skimmed,						16,341

On the fourth skimming 200 lb. of lime was used to thicken the slag. It is apparent that, while the percentage of copper drawn off in this slag is not quite halved, the amount of lead is a quarter more, with gold and silver less; and it seems likely that more oxidizing and slagging would be of advantage in removing lead. The amount of arsenic and antimony in this slag appears to be very small, when compared with the previous slags, and is not what was expected, since, with the basic slag of lead and lime, more elimination was looked for.

Average Assay of Slag Skimmed.

Au. Oz. p. t.	Ag. Oz. p. t.	Cu. Per Cent.	Cu. Lb.	Pb. Per Cent.	Pb. Lb.	As. Per Cent.	As. Lb.	Sb. Per Cent.	Sb. Lb.
0.11	29.2	32.3	5,276	16.6	2,719	2.3	380	1.9	304

There was removed from the furnace in slag, 1.5 per cent. of the copper, 58.8 per cent. of the lead, 7.6 per cent. of the arsenic, and 19.8 per cent. of the antimony, in the pig charged.

Taking the rate of slagging of copper as unity, the impurities went into the slag at the following faster rates:

	Lead.	Arsenic.	Antimony.
First slag, . .	33.4	5.5	14.6
Second slag, . .	35.4	5.2	12.9
Third slag, . .	33.9	4.5	12.6
Fourth slag, . .	95.6	1.4	1.5

From these figures it is obvious that, in the last slag, arsenic and antimony were coming off at about the same rate as copper, and that no more could be hoped to be removed through

slagging. As to lead, however, it appears that conditions were very favorable in this slag for a high extraction.

	Au. Oz. p. t.	Ag. Oz. p. t.	Cu. Per Cent.	Cu. Lb.	Pb. Per Cent.	Pb. Lb.	As. Per Cent.	As. Lb.	Sb. Per Cent.	Sb. Lb.	Lb.
Total tapped from the furnace,											333,765
	1.81	333.4	96.0	320,414	0.26	868	1.15	3,838	6.32	1,068	
Copper unaccounted for and assumed to have been left in the furnace,					.....	.....	.....	.....	.....	.....	13,431
Equivalent to anode-copper (at 96.0),					.....	.....	.....	.....	.....	.....	13,991
Containing at the above assay,					36	.....	161	.....	45		
Total impurities accounted for,					3,623	.....	4,379	.....	1,417		

The difference between this and the amount charged must have been driven off as fume while melting and working, as follows:

Pb. Lb.	Pb. Per Cent.	As. Lb.	As. Per Cent.	Sb. Lb.	Sb. Per Cent.
996	21.5	598	12.0	119	7.7

of the total in the pig charged.

There was therefore removed in furnace-refining 1.5 per cent. of the copper, 80.4 per cent. of the lead, 19.6 per cent. of the arsenic, and 27.5 per cent. of the antimony contained in the original pig charged, and, of the total impurities removed, 26.8 per cent. of the lead, 61.2 per cent. of the arsenic, and 28.1 per cent. of the antimony went off in the fume.

The results of these two runs check fairly well in the main. Both charges were about alike in impurities, excepting that charge B carried more lead. This was entirely accidental, since the pig was charged without any selection and represents conditions exactly as they existed. For comparison, the analyses of the two charges are set opposite each other in the following figures:

Average analysis of the copper as charged:

	Cu. Per Cent.	Pb. Per Cent.	As. Per Cent.	Sb. Per Cent.
Charge A,	93.9	0.676	1.38	0.40
Charge B,	93.1	1.27	1.37	0.42

Average slag-analysis:

	Cu. Per Cent.	Pb. Per Cent.	As. Per Cent.	Sb. Per Cent.
Charge A,	39.7	9.7	2.3	3.4 (?)
Charge B,	32.3	16.6	2.3	1.9

Per cent. of impurities removed from furnace in slag:

	Cu. Per Cent.	Pb. Per Cent.	As. Per Cent.	Sb. Per Cent.
Charge A, . .	1.97	67.1	7.7	39.6 (?)
Charge B, . .	1.5	58.8	7.6	19.8

Analysis of the anodes cast:

	Cu. Per Cent.	Pb. Per Cent.	As. Per Cent.	Sb. Per Cent.
Charge A, . .	96.36	0.15	1.15	0.33
Charge B, . .	96.0	0.26	1.15	0.32

Per cent. of impurities removed which went off in fume:

	Pb. Per Cent.	As. Per Cent.	Sb. Per Cent.
Charge A, . . .	11.82	12.82	?
Charge B, . . .	21.50	12.10	7.4

Extracted in furnace-refining:

	Cu. Per Cent.	Pb. Per Cent.	As. Per Cent.	Sb. Per Cent.
Charge A, . .	1.97	78.9	20.6	?
Charge B, . .	1.5	80.4	19.60	27.5

It will be noted that the refining has made the copper about 2.5 per cent. higher.

Comparison of slagging-coefficients (that of copper being assumed as unity):

	Lead.		Arsenic.		Antimony.	
Charge.	A.	B.	A.	B.	A.	B.
First slag, . .	28.0	33.4	3.5	5.5	17.3?	14.7
Second slag, . .	36.6	35.3	4.1	5.2	20.4?	12.9
Third slag, . .	37.2	34.0	4.3	4.5	20.2?	12.7
Fourth slag, . .	.....	95.4	....	1.3	.....	2.9

The slagging-rates are seen to hold about the same for any particular slag, but to decrease as the proportion of impurities decreases. Undoubtedly more slagging would have given a higher extraction of lead.

The greater volatilization of lead in charge B is undoubtedly due to more lead in the pig charged.

The above results are mainly of metallurgical, rather than of commercial, interest, since the quality of cathodes produced from this class of anodes does not vary noticeably when the fire-refining is carried further. While it would be interesting to learn the maximum extent to which lead, arsenic, and anti-

mony could be eliminated in such operations, the subsequent results achieved in the tanks did not seem to warrant the extra cost for fuel, furnace-repairs, danger to the furnace-bottom, extra labor, etc., involved in experiments for that purpose.

II. Electrolytic Refining.

The Influence of Soluble vs. Insoluble Impurities on the Purity of Electrolytic Copper.

The preceding section gives the results of fire-refining operations on very impure copper-bullion, undertaken in a 150-ton, Welsh type, reverberatory refining-furnace, in the endeavor to improve tank-house conditions for the electrolytic refining of this copper. In the present section, I hope to show that the impurities which are sometimes present in cast, electrolytically refined copper, in amount sufficient to impair seriously its conductivity, are there, not through electrolytic deposition, but entirely through the mechanical settling on the cathode, during its deposition, of suspended slime.

The data here presented are based upon observations of the working of the above-mentioned copper in the commercial tanks, both with and without much previous fire-refining, and comparisons drawn between this and the more usual grade of copper-bullion.

These results, taken together with earlier observations on floating slimes, have confirmed the belief, long entertained by me, that, with soluble anodes, it is the insoluble impurities of the slime, mechanically deposited during the cathode-growth, and not (as is generally believed) the soluble impurities of the electrolyte, electrolytically deposited, that cause the low conductivity of cast electrolytic copper, when the cathodes are deposited in an electrolyte of sulphuric acid, copper sulphate, arsenic, and antimony, of the degree of concentration usual in present practice.

In my opinion, electrolytes much more impure than is the present practice can be used with high-grade cathodes as a product. As will be seen from the figures following, good copper has been commercially made from electrolyte running as high in arsenic as 17.0 g. per liter, with copper sulphate at 144 g. per liter.

It is probable that most copper-refiners would believe the end at hand were they forced to use such an electrolyte; nevertheless, I noted no phenomena in connection with its use that would lead me to believe that the allowable limit of soluble arsenic was anywhere in sight. I do not mention antimony, since it can be shown that the electrolyte is soon saturated with that element, which thereafter goes entirely into the slime.

These experiments were discontinued early in 1910, when I resigned to take a position with another company; but I think the following record of them will establish my proposition.

From the well-known fact that slimes are high in impurities, it follows that a small amount of slime, settling on, and subsequently remelted with, the cathode, might add enough impurity to give a low conductivity in the resulting cast-copper.

We all know that arsenic is electrolytically deposited on the cathode from a solution depleted of copper sulphate; but an electrolyte thus depleted in copper sulphate should not be present in the commercial tank of a copper-refinery. This reaction, therefore, does not affect the present argument, except through the circumstance that copper-refiners may have different views as to what constitutes such a depleted solution; in other words, what is the lowest amount of copper sulphate in the electrolyte which will still give a good cathode-deposit.

With regard to this question, I think a depleted solution such as I have mentioned is one in which the concentration of copper ions is not sufficient to convey all the current to the cathode, under the existing conditions of current-density and solution-circulation. In that case, the current will seize upon the element present in the solution which stands next in the electro-chemical series. This happens to be arsenic; and the resulting deposition of arsenic in the cathode will give to the latter a lower conductivity. But it is not my present purpose to consider such a case. My argument proceeds on the assumption that the electrolyte is not depleted of copper.

Comparing the ratio of silver to arsenic and antimony in the average slimes, with the same ratio in the copper cast from cathodes deposited during the time the slimes were made, we find that this ratio is greater in the cast product. Heretofore, the inference has been that this additional amount of impuri-

ties must have been electrolytically deposited from the solution, and consequently that the greater the quantity of soluble impurities in the electrolyte, the greater their liability to deposition in the cathode. This conclusion is plausible; but, as I shall show, the solutions usual in copper-refining are relatively so pure that their impurities are a long way from beginning to deposit, and, therefore, measures taken to prevent their deposition are unnecessary as well as costly.

It is customary among copper-refiners, when the arsenic in solution rises seriously above 10 g. per liter, and it is impossible at the time to remove sufficient electrolyte to reduce it, to increase the proportion of copper sulphate over that usually carried, on the theory that the arsenic will then be less liable to deposit, and hence the conductivity of the cast-copper will be maintained. This also is a natural inference; but whether a higher concentration of copper sulphate is actually necessary or not, will depend on how nearly the electrolyte is "depleted" in this salt.

In my judgment, this increase of copper sulphate is, in general, not only unnecessary but positively disadvantageous, for the reasons: (1) that a further increase of copper sulphate increases the specific gravity of the solution, thereby giving slimes more chance to float to the cathode; (2) that, the higher the concentration of the copper sulphate, the more it acts as an oxygen-carrier, thereby aiding the acid in dissolving, chemically, more copper from the anode, with a consequent more rapid increase of the soluble copper in solution. To keep this amount within bounds requires either the removal of a part of the solution for the crystallizing-out of its copper, or the installation of insoluble anodes for the deposition of this excess of copper (both of these operations are costly, unless, in the first case, bluestone is made for the market, and even then there is liable to be a loss at present prices of that article); (3) that, as I believe, though I have not been able to prove it to my own satisfaction, a high copper-sulphate concentration tends to cause polarization on the anodes, thereby giving a larger amount of floating slime; and (4) that high copper sulphate in the electrolyte tends to cause a very coarse crystallization in the cathode-growth, thereby giving plenty of spaces for holding floating slime.

In testing the hypothesis that the greater proportion of arsenic and antimony to silver in the cast copper, as compared with the corresponding proportion in the slime, was due to the slime only, and not to electrolytical deposition, settled slime was collected at various points of the solution-circulation.

For the purpose of reducing all results to a comparable basis, the amount of arsenic and antimony per ton of slime has been compared in each case with that of the silver existing there, as so many avoirdupois ounces of arsenic or antimony to one troy ounce of silver, because silver is the only large and totally insoluble constituent of the slime.

The average analysis, for 15 weeks, of weekly composite regular slimes swept from the tanks shows:

Au. Oz. p. t.	Ag. Oz. p. t.	As. Per Cent.	Sb. Per Cent.
146.2	13,569	3.2	6.0

Ratio of gold to silver: 1 oz. Au to 93.0 oz. Ag.

Ratio of silver to arsenic and antimony: 1 oz. Ag to 0.076 avoir. oz. As, and 0.142 avoir. oz. Sb.

Slime which had settled in the return-solution launders, just outside the tanks, ran:

Au. Oz. p. t.	Ag. Oz. p. t.	As. Per Cent.	Sb. Per Cent.
54.93	6,130	5.3	13.0

Ratio of gold to silver: 1 oz. Au to 112 oz. Ag.

Ratio of silver to arsenic and antimony: 1 oz. Ag to 0.277 avoir. oz. As, and 0.680 avoir. oz. Sb.

Slimes collecting in a lead-fiber filter, just ahead of the pumps, gave:

Ratio of gold to silver: 1 oz. Au to 118 oz. Ag.

Ratio of silver to arsenic and antimony: 1 oz. Ag to 1.40 avoir. oz. As, and 1.34 avoir. oz. Sb.

Slimes escaping the filter and settling beyond it, but before reaching the pumps, ran:

Ag. Oz. p. t.	As. Per Cent.	Sb. Per Cent.
3,235.6	8.0	26.5

Ratio of silver to arsenic and antimony: 1 oz. Ag to 0.794 avoir. oz. As, and 2,620 avoir. oz. Sb.

Slimes settling in the section-boxes, just before the solution entered the first electrolytic tanks on its return, and therefore representing the lightest float-slime that it was possible to catch—since it had completed the solution cycle—gave:

Au. Oz. p. t.	Ag. Oz. p. t.	As. Per Cent.	Sb. Per Cent.
71.57	471.0	16.0	33.8

Ratio of gold to silver: 1 oz. Au to 6.6 oz. Ag.

Ratio of silver to arsenic and antimony: 1 oz. Ag to 10.87 avoir. oz. As, and 22.91 avoir. oz. Sb.

From these figures it appears that the longer the slime floats, the higher the amount of arsenic and antimony it contains, running from 0.076 avoir. oz. As and 0.142 avoir. oz. Sb per troy oz. silver in the slime falling in the tanks, to 10.87 avoir. oz. As and 22.91 avoir. oz. Sb per troy oz. Ag in the lightest float, or an increase of arsenic of 142 times, and of antimony of 161 times, that in the main bulk of slime for the same silver-content. (These figures may serve also to give some idea of the degree of fineness of the gold-particles, since, in the slime floating longest, there is, for the same silver-content, five times as much gold as in the main bulk of the slime, notwithstanding the fact that gold is 1.83 times as heavy as silver.)

Naturally, the amount of slime-float passing any given point is smaller the farther this point is from the source of production. Hence it would be entirely in order to assume that if mechanically-deposited slime, instead of the solubles in the electrolyte, were responsible for the copper-refiner's troubles, the excess of impurity over silver in the cast product, above that in the main bulk of the slime, should be accounted for by the impurity in the first float-slime caught. Moreover, since the main bulk of the slime must have fallen directly from the anodes to the bottom of the tanks, it could not rightly be classed as float, and could not have remained suspended in the solution long enough to settle in any great amount on the cathode.

If these assumptions are correct, we may naturally expect to find the ratio of silver to arsenic and antimony in the first float similar to that in the cast product. Within the reasonable limits of error to be expected in dealing with lots of com-

mercial size, and with small percentages of impurity, this conclusion is confirmed by the following figures.

The average analysis of the wire-bar cast during a period of three months shows gold, silver, arsenic, and antimony as follows:

Au. Oz. p. t.	Ag. Oz. p. t.	As. Per Cent.	Sb. Per Cent.
0.0116	1.833	0.00162	0.00171

Ratio of gold to silver: 1 oz. Au to 110 oz. Ag.

Ratio of silver to arsenic and antimony: 1 oz. Ag to 0.282 avoir. oz. As, and 0.298 avoir. oz. Sb.

These figures compare very closely with the ratios obtained from the analyses of the first float-slime; and, ignoring the consideration of what lighter and more impure float might be caught on the cathode before leaving the tank, the impurities in the wire-bar can be entirely accounted for by the first and heaviest float. I may explain here that by the word "float" I mean the slime either floating on or in the solution, as distinguished from the slime which settles directly, upon being detached from the anode.

While the figures given above seemed to establish my hypothesis, they did not furnish answers to all objections. But, about a year later, further opportunity of investigation was given by the very impure copper-bullion mentioned at the beginning of this paper.

For a long time, this bullion had been treated by mixing it in small proportion with purer copper; but the results were not at all satisfactory, and the resulting cast-copper was liable to be low in conductivity. In trying to overcome this evil, and starting from the foregoing hypothesis, that the insoluble slime, rather than the soluble constituents of the electrolyte, was the source of the final trouble, I had this impure pig charged by itself into the anode-furnace, with the purpose of refining it there as far as possible, and then further refining it electrolytically in a separate electrolyte. I reasoned that, treating this bullion by itself, we would get better cathodes from the usual grade of anodes, and that the cathodes made from the leady anodes could either be cast into separate ingots of inferior quality, or mixed before casting, in predetermined proportion, with the material furnished by the better cathodes.

As the experiment had to be finally performed, there was only enough of this impure copper to supply about three-fourths of the tanks set aside for this separate circulation. The rest had to be loaded with the purer anodes. Yet, in spite of this interference of commercial requirements with purely scientific investigation, the results were significant.

As a consequence of the large proportion of very impure copper present, the electrolyte increased very rapidly in arsenic, in spite of large daily removals of solution, until, at the time the experiments were stopped, it held 17 g. of arsenic per liter. Yet, in every case, the copper cast from the cathodes taken from tanks loaded with good anodes was above the standard in conductivity. In appearance, these cathodes were in all cases similar to those deposited in other and purer solutions from the same grade of anodes. But cathodes taken from the tanks loaded with leady anodes were very dirty with slime, and, even when washed and carefully scrubbed, were only fit for low-conductivity ingots.

Two furnace-charges of cathodes deposited from good anodes in high-arsenic solution were taken for observation. They were in every way representative of all the others, and therefore will serve as examples. They will be designated as charge X and charge Y.

Charge X contained 64 per cent. of the above class of cathodes, the balance being cathodes from the same class of anodes, deposited in a solution carrying about 9 g. of arsenic per liter.

Test-wires for conductivity taken from this charge after the 1st, 3d, and 5th sixths had been cast, gave 97.3, 97.7, and 97.0 per cent. of conductivity, Mathiessen standard, hard drawn. The silver in the cast copper was 1 oz. per ton. The average analysis of the electrolyte in grams per liter, during the two weeks in which this copper was depositing, was

Sp. Gr.	Free Acid.	Copper Sulphate.	As.	Sb.	Cl.	Ni.	Fe.
1.210	123	133	15.3	0.50	0.025	17.9	2.2

The anodes from which this 64 per cent. was deposited analyzed:

Au, 1.76 oz. p. t.; Ag, 288.0 oz. p. t.; Pb, 0.05; Cu, 97.92; As, 0.16; and Sb, 0.19 per cent.

Charge Y contained 72 per cent. of this class of copper; and

wires drawn from samples taken as above ran 97.3, 97.4, and 97.4 per cent., respectively, Mathiessen standard, hard drawn. The silver in the cast copper was 0.8 oz. per ton. The average analysis of the electrolyte in grams per liter, during the two weeks in which this copper was depositing, was

Sp. Gr.	Free Acid.	Copper Sulphate.	As.	Sb.	Cl.	Ni.	Fe.
1.215	109	144	17.0	0.50	0.026	17.9	2.20

The anodes making this 72 per cent. closely approximated those of charge X.

The balance of the charge was of cathodes from the same grade of anodes, but deposited in a solution averaging in the neighborhood of 7.5 g. of arsenic and 12.3 g. of nickel per liter, the other salts being about the same as in the above solution.

Undoubtedly it would have been better to make the above charge entirely of cathodes deposited in the more impure solution; but this was not possible at the time, since the furnace-charge was 125 tons, and operating-conditions would not permit us to hold this copper back. In general appearance, however, there was no difference between the cathodes deposited in the different solutions.

From the above figures it would appear that the soluble impurities in the electrolyte, in amounts up to those last shown, do not affect the purity of the deposited copper to any greater extent than do those in a purer electrolyte, since the conductivity of the copper cast from cathodes deposited in an impure solution was in no way inferior to that of copper cast from cathodes deposited in a purer electrolyte from anodes of the same grades.

The following results also confirm the theory, in so far as it has been tested with impurities to the amount above given.

When cathodes were being taken from a section of 34 tanks which had been loaded with the impure, leady anodes, a cathode was picked out which was a very good representative of all the rest. All the cathodes were extremely dirty.

A wire directly drawn from this representative cathode gave a conductivity of 99.3 per cent., Mathiessen standard, hard drawn, and a tensile strength of 61,200 lb. per sq. inch.

The average analysis of the electrolyte in grams per liter, during the time this copper was depositing, gave:

Sp. Gr.	Free Acid.	Copper Sulphate.	As.	Sb.	Cl.	Ni.	Fe.
1.210	113	131	16.6	0.50	0.022	17.9	2.2

Had any of this arsenic been deposited electrolytically on the cathode, it would be in such combination as to affect the conductivity of the wire very considerably. This was not the case.

The entire lot of these cathodes was cast into ingots, forming 97 per cent. of the charge so cast. The conductivity of the copper, as given by samples taken after casting the 1st, 3d, and 5th sixths of the charge, was 94.4, 94.9, and 94.4 per cent., Mathiessen standard, hard drawn. The copper plus silver in the ingots was 99.946 per cent.; and the silver was 3.8 oz. per ton.

It should be remarked that, before charging into the furnace, these cathodes received a thorough scrubbing with brush and water, and were treated in the furnace so as to remove as much of their impurities as possible.

Analyses were made as follows:

	Au. Oz. p. t.	Ag. Oz. p. t.	Cu. Per Cent.	Pb. Per Cent.	As. Per Cent.	Sb. Per Cent.	Ni. Per Cent.
Ingots.....	.....	3.8	99.946	.....	0.0040	0.0048	0.0046
Cathode...	.....	0.9	.....	.....	0.0016	0.0019	0.0028
Slimes.....	25.8	5434.2	8.6	22.5	6.8	11.8	5.8
Anodes...	0.83	213.9	96.59	0.65	1.00	0.32	0.2044

The anode-sample was a composite of two samples taken during the time the anodes were cast; the slimes-sample was taken from the same tank as the cathode.

If arsenic, or other impurity, in the electrolyte, had been deposited electrolytically on the cathode, all portions of the latter should have been equally affected. Working on this basis, and endeavoring to eliminate, as far as possible, all superficially-adhering slime, we cut out a piece near the top of the cathode and about 5 by 8 in. in size, and thoroughly washed the surface with dilute nitric and then with dilute sulphuric acid. This should leave in the sample only: (1) those impurities which had mechanically settled on the cathode during its deposition, and had been covered by copper subsequently deposited; and (2), the impurities electrolytically deposited from the electrolyte. The piece was sawed to destruction, and the sawdust was taken for analysis. The result of this analysis is given above under the heading "Cathode."

Since there is no soluble silver in the electrolyte, all silver found in the cathode must have come from settled slime; and

on the supposition that all slime-impurities followed the silver mechanically on to the cathode, in the same ratio as that presented in the slime, the above cathode-analysis would show as follows:

	Ag. Oz. p. t.	As. Per Cent.	Sb. Per Cent.	Ni. Per Cent.
Cathode, . .	0.9	0.00113	0.00195	0.00096

Especially as regards arsenic and antimony, these figures check very closely with the actual analysis, even though the calculated analysis does not take into account the influence of the heavy float, which must have been considerable in amount, as shown by the extremely dirty condition of the cathodes. The influence of such float has been shown in a former part of this paper. It is evident that little, if any, room is left for the recognition of electrolytically deposited arsenic or antimony.

The foregoing results show, I think, that such impurities as are found in cast electrolytic copper are due to mechanically-held slime and not to impurities electrolytically deposited from solution.

It is when impurities from alloys are formed with the copper that they affect its conductivity; and my experiments with the cathode show that such alloys must be made subsequently in the ingot-melting furnace, by the chemical union, through fusion, of the mechanically-held slime with the copper of the cathode, and that they were not made by electro-deposition from the solution.

How high the arsenic may be allowed to go in solution in the electrolyte (copper sulphate and acid remaining the same), before it will begin to be deposited, I am not prepared to say. It seems, however, that arsenic does not begin to be deposited up to 17 g. per liter with copper sulphate in the neighborhood of 140 g. per liter; and it is probable that the allowable amount of soluble arsenic may be increased very considerably over that now carried in usual commercial electrolytes, without running any risk of impairing the conductivity of the product.

It must also be borne in mind, however, that the slimes must be kept off the cathode, and that the less they settle there the purer will be the resulting copper.

In present practice, when there is anything wrong with the conductivity of the copper, it seems to be usual to seek a remedy by altering the electrolyte. This appears to me to be

looking in the wrong place for the evil to be remedied. In my judgment, it is generally the preliminary refining, and the consequent quality, of the anode which chiefly need attention. These adjusted properly, the slimes will be heavy and will not cling to the anode; and, as a consequence, a higher percentage of impurity will fall at once out of the metallurgical calculation; there will be less "float"; and the result will be purer cathodes. I am thoroughly convinced that, in comparison with these requirements as to the anode, there is a great leeway in the composition of the electrolyte.

SUMMARY.

1. The impurities in cast electrolytic copper are present in about the same proportion as they exist in the heaviest float-slime, and are there because of that float.

2. The conductivity of copper cast from cathodes deposited from good average anodes in an electrolyte containing arsenic in amount up to 17 g. per liter, is as good as that of copper cast from cathodes deposited from the same grade of anode in a solution containing not more than 8 g. of arsenic per liter.

3. The wire drawn from a cathode representing 34 tanks of dirty cathodes deposited from high-arsenic, leady anodes in a solution containing 17 g. of arsenic per liter, gave a conductivity, hard drawn, of over 99 per cent., Mathiessen standard, whereas these same cathodes melted and refined in the furnace gave a conductivity, on the same basis, of less than 95 per cent.

4. The analysis of impurities known to be held wholly in the cathode—*i. e.*, none of which adhered to the surface—has shown that they were present in practically the same proportions as they exhibited in the slimes, and therefore must be due to mechanically-held "float." It follows that the copper itself must have been deposited as pure copper; and this is confirmed by my last paragraph, since only very pure copper would give a hard-drawn conductivity-test of over 99 per cent., Mathiessen standard.

5. When the percentage of anode-impurities going to slime is calculated, some interesting facts appear. According to the figures given above, all the lead, 26.7 per cent. of the arsenic, all the antimony and all the nickel went to the slimes. That this is the usual thing with anodes of this class, can readily be shown. The natural expectation would be that the nickel

would go into solution as nickel sulphate; and so it does, to a certain extent. Its appearance in the slime can very probably be accounted for by the fact that nickel and arsenic have a considerable affinity for each other, and form a speiss which is insoluble to the electrolytic corrosion. The surplus of nickel does go into the solution; and it is well known that practically all commercial electrolytes carry nickel.

The large difference shown between the calculated and the observed values of nickel in the cathode, is a mystery which I have been prevented from investigating.

Treatment of Mine-Water from the Ashio Copper-Mine.

BY JOSEPH W. RICHARDS,* SOUTH BETHLEHEM, PA.

(New York Meeting, February, 1912.)

THE Ashio copper-mine of the Furukawa Mining Co. is situated 18 miles from Nikko, and 109 miles north of Tokyo, near the center of Japan. The mine-waters are run over scrap-iron, whereby most of the copper is precipitated as cement-copper, leaving iron sulphate with some copper sulphate in solution. After using in the wet concentrating ore-dressing plant, the waste water contains clay and ore-slimes. The only avenue of disposal of this water is the Watarese river; but as this flows through valleys in which the water is used for irrigating the rice-crops, it is necessary to purify and clarify the water before discharge.

The water amounts to from 600 to 700 cu. ft. (from 17 to 20 cu. m.) per minute, and averages about 0.00028 per cent. of copper, representing more than 18 tons of copper per month. The water carries H_2SO_4, $FeSO_4$, $Fe_2(SO_4)_3$, $CuSO_4$ and other metallic sulphates in solution, and clay, ore-fines, slimes and basic iron sulphates as suspended matters. To purify and clarify the water, precipitation by milk of lime has been used, in three different ways, as follows:

I. *First Method, Copper not Recovered.*

Milk of lime was run into the water, and the mixture run through six settling-ponds, and then through a sand filter into

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the river. Fig. 1 is a view of the settling- and filtering-ponds. In the first pond, sand, metallic hydrates, and some slime collected; in the next five ponds, slime, calcium sulphate, and metallic hydrates carrying copper settled out. The effluent carried from 0.1 to 0.05 mg. of copper per liter (from 0.00001 to 0.000005 per cent.), indicating the precipitation of from 96 to 98 per cent. of the copper present in the water treated. The sand, slime, and metallic hydrates from the first pond were worthless; and although the precipitate taken from the other ponds carried from 1 to 1.5 per cent. of copper, yet the presence of about 25 per cent. of alumina and 60 per cent. of silica in it (caught by the flocculent hydrate precipi-

FIG. 1.—SETTLING- AND FILTERING-PONDS OF THE FURUKAWA MINING CO., ASHIO.

tate) rendered it altogether useless for smelting, and it was removed to the dump. The 18 or 19 tons of copper per month in these waters was thus lost.

II. *Improved Method, to Enrich Precipitate.*

The water was first allowed to run through four settling-ponds, to coagulate and then to settle the slime and other fine matters suspended colloidally by the electrolytes such as H_2SO_4 and sulphates present in the mine-water. The first pond gathered sand and some slime, while in the next three were deposited some slime and fine matters, which were coagulated, carrying from 0.5 to 1.0 per cent. of copper, and more than 60 per cent. of silica—a worthless material. In the fifth pond milk of lime was added, producing in this and the sixth pond

a deposit consisting chiefly of metallic hydrates and some calcium sulphate, but still mixed with considerable slime. This precipitate averaged, copper, 4; iron oxide, 22; alumina, more than 20; and silica, more than 30 per cent. While this precipitate was richer than that previously obtained, yet it was still too high in alumina and silica to pay to smelt it, and further effort was directed towards obtaining a richer copper-precipitate.

III. *Best Method, Saving the Copper.*

The water is run alone through three settling-ponds, in the first of which sand and some slime settle, while in the second and third some coagulated matters settle, carrying some copper and slime high in silica, as explained in method II. In the fourth pond milk of lime is run in in small quantity, from 10 to 20 per cent. of the whole amount required for complete neutralization and precipitation. This causes a precipitate to settle, which is mostly hydrated iron oxide and low in copper, since there is from five to six times as much iron in the solution as copper; the flocculent precipitate also carries down with it most of the remaining colloidal slime in suspension, consisting mostly of alumina and silica. The water flows off into the fifth pond, where the rest of the lime necessary for the treatment is added. This precipitates the metallic hydrates completely, the precipitate settling in this pond and the next. The overflow from the sixth pond is passed through a sand filter before discharge, as before. Table I. presents three analyses of the precipitate recovered from tanks Nos. 5 and 6, which is dried and sent to the smelter, being rich enough in copper and low enough in alumina and silica to pay for the expense of treatment. The last column in this table gives the rational composition of the precipitate, calculated from the analysis of sample No. 3.

TABLE I.—*Analyses of Precipitate Dried at* 110° *C.*

	Sample No. 1. Per Cent.	Sample No. 2. Per Cent.	Sample No. 3. Per Cent.	Rational Composition of Sample No. 3.	Per Cent.
Cu	5.04	5.64	6.65	$CuO.H_2O$ . .	10.18
Fe	11.93	12.67	14.14	$Fe_2O_3.3H_2O$ . .	27.02
CaO	5.35	6.51	8.42	$CaSO_4.2H_2O$ . .	25.86
Al_2O_3	4.92	4.90	3.28	$Al_2O_3.3H_2O$ . .	4.93
Mn	3.85	4.66	3.70	$Mn_2O_3.3H_2O$ . .	7.13
SiO_2	9.74	12.34	11.68	$SiO_2.2H_2O$ . .	18.69
Loss on ignition, .	30.28	31.08	31.34	Moisture, . . .	4.88
					98.69

The method as thus improved not only saves 18 tons of copper per month, which was formerly lost as a worthless precipitate on the dump, but also gives clear waste water not injurious to agriculture.

In conclusion, I wish to thank the Furukawa Mining Co. for permission to make public this method of fractional precipitation (which is not covered by any patents, and is therefore free to the public), and to express my indebtedness to I. M. C. Imai, engineer of the company, for furnishing the notes on which this paper is based.

Vanadium in Pig-Iron.

BY PORTER W. SHIMER, EASTON, PA.

(New York Meeting, February, 1912.)

It does not seem to be generally known that some American pig-irons contain notable amounts of vanadium, and while the present investigation is far from covering the whole range of irons, it is hoped that, at least, it may serve to call attention to the importance of looking for this element. The majority of the pig-irons investigated were made in the Lehigh Valley and neighboring furnaces, using more or less New Jersey magnetite in their mixtures. Many of these ores, and probably all of them, contain a little vanadium as well as titanium; the ore from one of the largest of these mines contains 0.05 per cent. of vanadium and 0.60 per cent. of titanium.

Dr. Thomas M. Drown was the first to detect the presence of vanadium in the incrustations[1] occurring on certain pig-irons, and he also found this element in a number of magnetites, but the methods for its accurate determination had not, at that time, been well worked out. In a general way, it may be said that the Lehigh Valley pig-irons contain from 0.02 to 0.05 per cent. of vanadium and from 0.10 to 0.20 per cent. of titanium.

These elements follow the silicon; that is, the furnace-conditions which produce a high-silicon, low-sulphur iron will cause the maximum reduction of vanadium and titanium, and

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[1] Robertson and Firmstone, Note Concerning Certain Incrustations on Pig Iron, *Trans.*, xii., 641 (1883–84).

the lower grades of iron, with low silicon and high sulphur, have a lower content of these elements. The following analyses of irons made from the same mixture, but under varying furnace-conditions, show this very well:

	Vanadium. Per Cent.	Silicon. Per Cent.	Sulphur. Per Cent.
No. 1 X, . . .	0.042	3.46	0.022
No. 2 plain, . .	0.038	2.26	0.041
Mottled, . . .	0.029	0.52	0.106

The following are the results of vanadium-determinations in a number of pig-irons made in the Lehigh valley and in New Jersey:

	Vanadium. Per Cent.
Sample A.	0.028
Sample B, ...	0.032
Sample C, ...	0.039
Sample D, ...	0.050
Sample E, ...	0.049
Sample F, ...	0.046

That not all pig-irons made wholly of magnetic ores contain these notable amounts of vanadium is shown by the fact that two widely-separated furnaces, using all magnetite, none of which is from New Jersey, were found to contain respectively 0.006 and 0.012 per cent. of vanadium. An iron made wholly of Virginia hematite ore contains 0.014 per cent. of vanadium, and two different irons, presumably made wholly from Lake Superior ores, contain respectively 0.009 and 0.023 per cent. Vanadium sometimes occurs in limestone and in the ash of coke, and these no doubt may contribute a part of the vanadium which finds its way into the iron.

Determinations of vanadium in iron castings made at different times and places in the Lehigh valley showed vanadium-contents as follows: 0.033, 0.013, 0.022, 0.024, and 0.027 per cent. To ascertain the loss of this element on remelting in the cupola, a mixture of three irons was used in which the vanadium had been carefully determined just before charging. The average content of these irons was 0.0313 per cent. and the average content of the castings was 0.027 per cent., showing a loss of 13.74 per cent. of vanadium on remelting.

There is reason to believe that the effect of vanadium on the

strength and other qualities of these irons is noticeably beneficial, and titanium probably acts in a similar way. When these elements are present in notable amounts there is often a peculiar fine-grained structure combined with softness and unusual strength. I have often seen pig-iron made wholly or in large part of New Jersey magnetites which showed this grain to a very marked degree. Such irons often present a peculiar mottled appearance well described by Bauerman in his description of titanium pig-iron as made up of "dull, dark gray patches set in a white net work." Superficially, the appearance is that of hard, mottled iron, but, in reality, the iron is very soft, even when the silicon is well below 1 per cent. The sulphur, also, is quite low in these irons. Sometimes the fracture of a pig-iron will show partly normal gray iron and partly the peculiar mottle; the lower half of the pig may be mottled and the upper half gray, or there may be simply a mottled spot surrounded by gray iron. It may be noted, however, that, without exception, the fine-grained mottled part of the pig is very noticeably softer than the gray part of the same pig. The drillings are curled and partake of a steely character. This iron also finishes beautifully in the lathe, and takes a high polish.

It has long been my view that the peculiar grain is due to vanadium or titanium, or to both together. The latter element, present in the iron as a carbide, is often as high as 0.30 per cent. Neither vanadium nor titanium seems to be alloyed with the iron; both are probably present as carbides; at all events, they are found almost wholly, if not entirely, in the insoluble graphitic residue left after the iron has been dissolved in very dilute hydrochloric acid. The vanadium compound is in an even finer state of division than the titanium carbide, which latter occurs in cubical crystals that can be readily separated from the insoluble residue. (See my paper, Titanium Carbide in Pig-Iron.[2])

Since neither vanadium nor titanium is alloyed with the iron, it may be asked how they can have any effect on its structure or strength. Apart from any cleansing action these elements may have by removing any nitrogen and oxygen that

[2] *Trans.*, xv., 455 (1886-87).

may be present, the suggestion is here ventured that they affect the structure by separating out earliest during the solidification of the iron as uniformly-distributed, finely-divided carbides, which furnish starting-points or nuclei for the rapid crystallization of the iron, in this way causing a finer grain—with its accompanying greater strength—than would be the case in the absence of these elements. That titanium carbide thus crystallizes out while the iron is still soft is proved by its perfect, sharp-edged cubical crystals.

In this connection, the results of a test of a vanadium-steel, containing 0.143 per cent. of vanadium, may be of interest. After dissolving a sample of the steel in dilute hydrochloric acid (25 cc. of HCl, 1.20 sp. gr., to 200 cc. of water), the black, carbonaceous residue was filtered off and in it was found 0.068 per cent., or nearly one half, of the total vanadium. On dissolving the steel in a cold solution of double chloride of copper and potassium the residue again contained 0.068 per cent. of vanadium. Thus, even in the standard vanadium-steel, about one-half of this element is present in an unalloyed form, probably as a carbide.

As to the addition of vanadium to pig-iron for the purpose of increasing its strength and refining its grain, it is probably safe to assume that greater improvement may be looked for when it is added to weak irons which do not already contain notable amounts of this element.

The Treatment of Complex Silver-Ore at the Lucky-Tiger Mine, El Tigre, Sonora, Mexico.

BY D. L. H. FORBES, TORONTO, CANADA.

(New York Meeting, February, 1912.)

Introduction.

THE application of the cyanide process to complex sulphide ores for the extraction of the gold and silver is no longer a novelty; but, as an example showing modern tendencies in this direction, the following description of the new milling process at the Lucky-Tiger mine is offered:

The extraordinary richness of the ore in the upper workings of this mine made any thought of milling in its early days seem unnecessary. Its location in an almost inaccessible part of the rugged mountains just south of the Arizona boundary-line, and 60 miles from the town of Douglas, made the expense of bringing in machinery and supplies a very serious consideration. But, after the property had been shipping high-grade ore on mule-back for a few years, there remained in the mine a considerable tonnage of vein-matter that was too low in value to ship, but which it was thought could be milled near the mine at a profit. The first milling-experiment consisted in the installation of Nissen stamps in conjunction with concentrating-tables and Frue vanners, but, as the ore from the upper workings contained most of its silver in the form of cerargyrite, this type of mill was not successful.

The proximity of the mine to the great copper-concentrating mills of Cananea and Moctezuma naturally made the influence of copper-milling ideas felt in the company's next installation. At considerable expense a stage-crushing concentrating-mill, with a capacity of 100 tons per day, was built. This mill was modeled along the lines of the most approved practice in copper-concentrators at that time, the main idea being that crushing by stages in rolls was necessary in order to prevent losses from "sliming." Largely owing to the fact that, when the mill

began operation, the mine produced ore with very little cerargyrite, and with the silver and gold intimately associated with such minerals as galena, zinc-blende, chalcopyrite, and iron pyrite, this method of concentration was profitable, in spite of the fact that the mill-tailings carried 0.1 oz. of gold and 15.0 oz. of silver per ton, and power for operating the machinery cost $275 per h-p-year.

In the early part of 1910 I installed a small experimental cyanide-plant for the purpose of testing the ore and mill-tailings. The results of experiments made in this plant were so satisfactory that it was decided to commence work at once on the construction of a 250-ton cyanide-plant, and to make such changes and additions to the existing works as would make possible the most economical milling of 175 tons of ore from the mine daily, in addition to 75 tons of old tailings from the dumps below the mill. This work was completed and the new process in operation by the end of June, 1911. A view of the mill and cyanide-plant is given in Fig. 1, and the general plan and sections in Figs. 2 and 3.

Character of the Ore.

The results of tests made in the experimental plant established the following facts in regard to the character of the ore:

(1) Very fine grinding is necessary in order to obtain a good extraction of the silver. Grinding samples of the tailings to pass a 150-mesh screen and other samples of the same material to pass a 200-mesh screen before cyanide-treatment gave results in favor of the finer grinding, showing an increase in extraction of from 15 to 20 per cent.

(2) A high decomposition of cyanide was to be expected, but, by careful concentration previous to treatment with solution, the consumption could be kept down to from 4 to 5 lb. of sodium cyanide per ton of tailings.

(3) Ore from certain sections of the property was found to be more amenable to cyanide-treatment than that from other places. In general, it was found that the oxidized ore, which yielded only a small recovery by concentration alone, gave the best extraction by combined concentration and cyanide-treatment of any ore in the mine. The cyanide-consumption on the oxidized ore was less than that on the sulphide ore. The worst

FIG. 1.—GENERAL VIEW OF MILL AND CYANIDE-PLANT OF THE TIGRE MINING CO., EL TIGRE, SONORA, MEXICO.

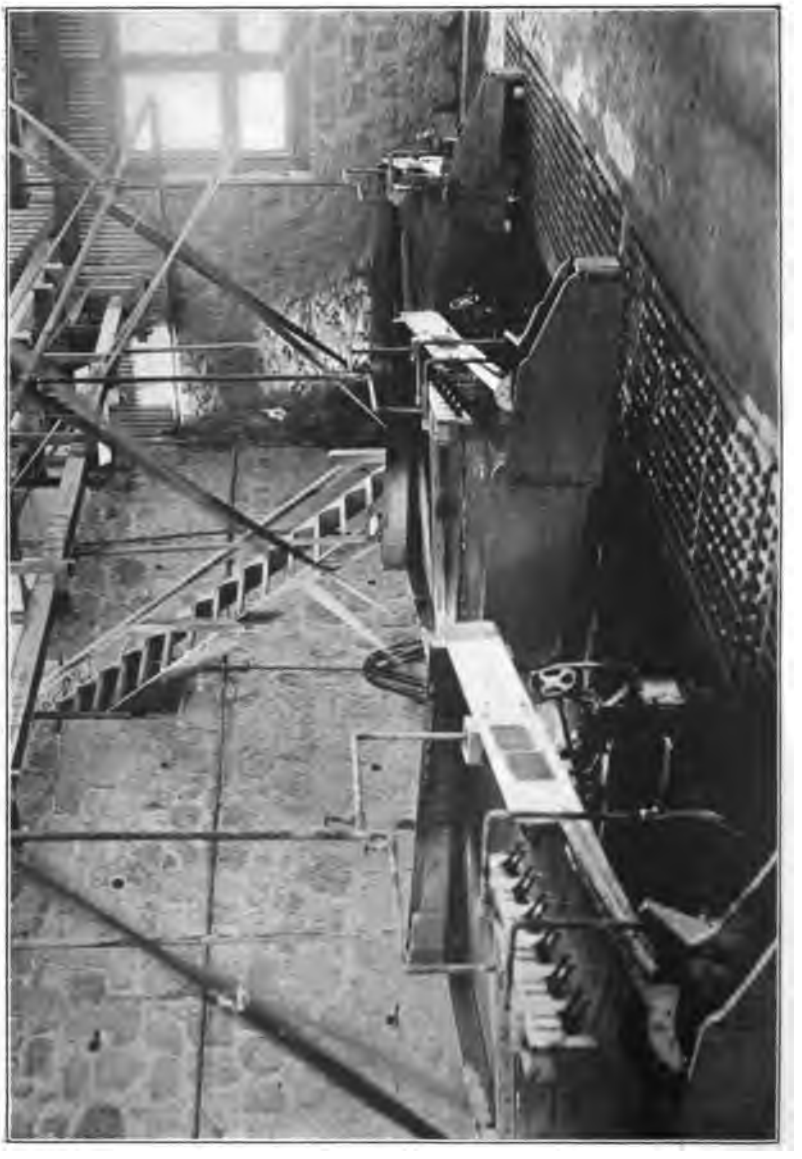


FIG. 6.—DEISTER SLIME-TABLES, NO. 2 MILL.

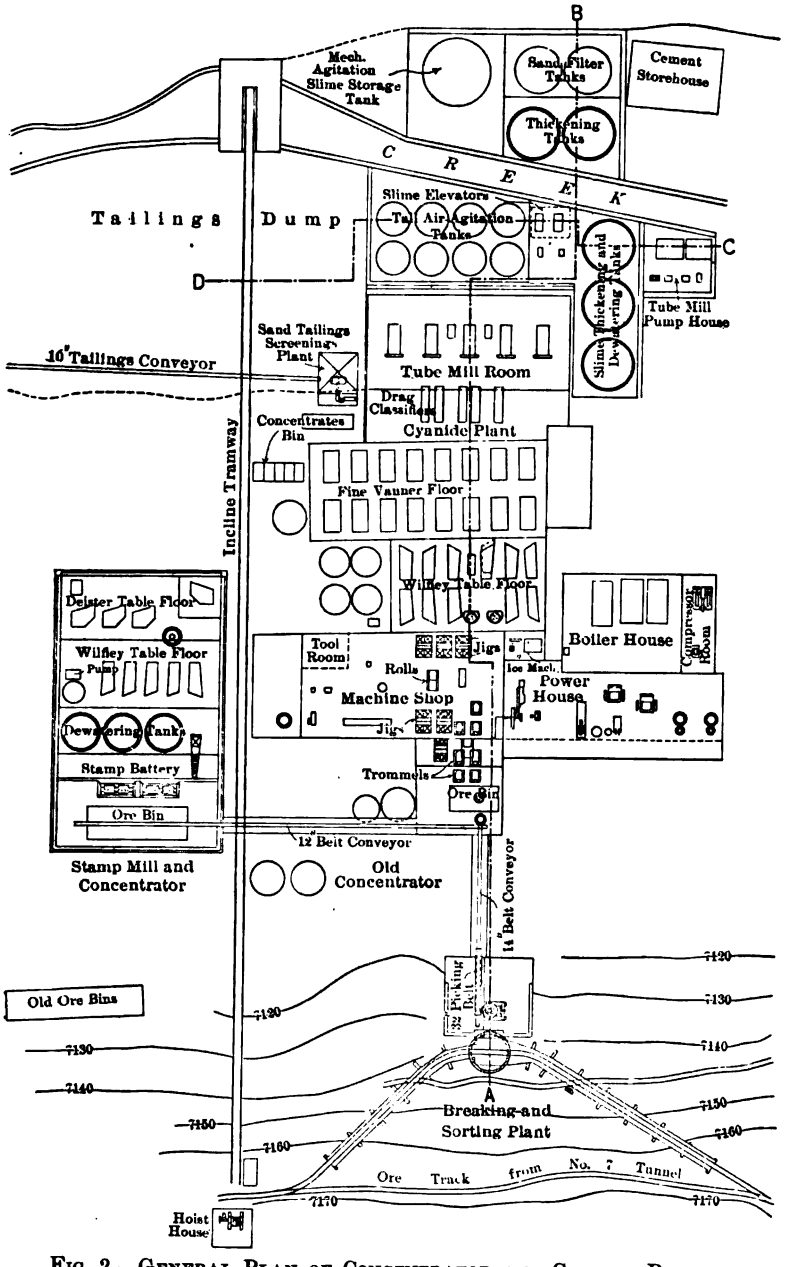


FIG. 2.—GENERAL PLAN OF CONCENTRATOR AND CYANIDE-PLANT.

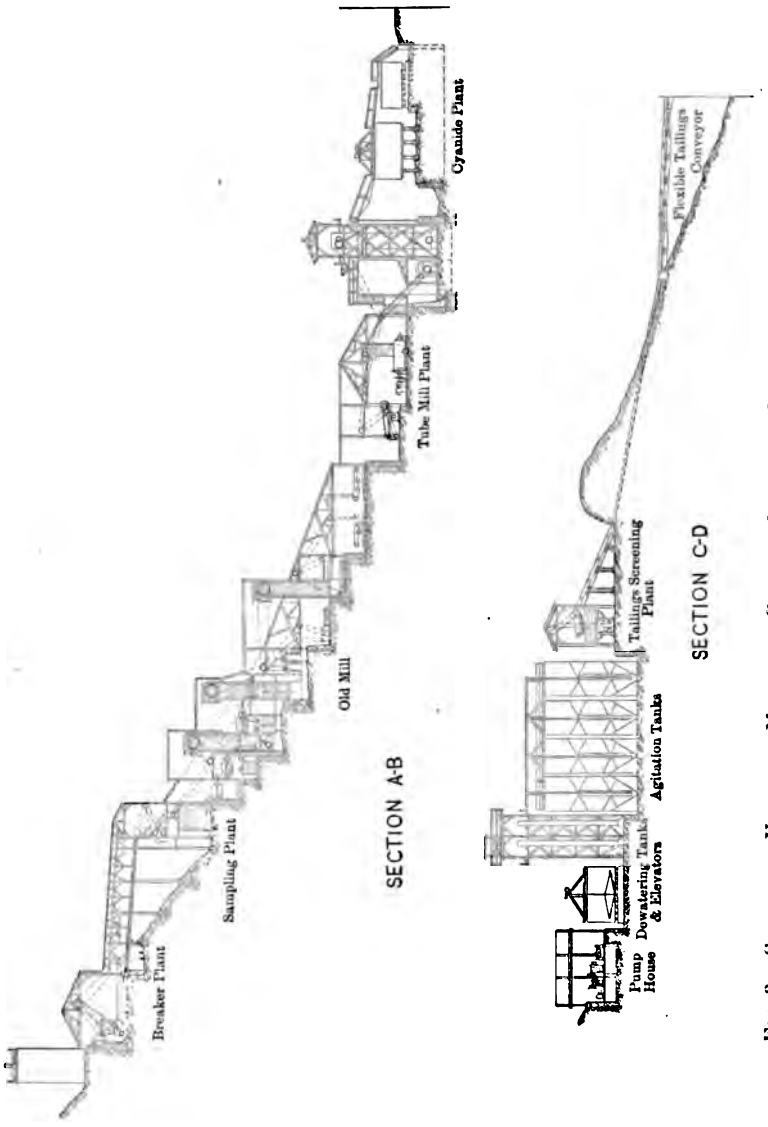


FIG. 3.—SECTIONAL VIEWS OF MILL AND CYANIDE-PLANT ON LINES A–B AND C–D OF FIG. 2.

results, both in the matter of cyanide-consumption and in extraction, were obtained from the tailings from a certain class of sulphide ore that contained a high percentage of copper and was associated with massive zinc-blende and galena. This class of ore comes from a vein known as the "Sooy." The best results were obtained with the tailings of ore from the southern end of the property in the Gold Hill mine. The ore from the latter place is partly oxidized and is comparatively low in copper.

The average assay of 38,610 tons of ore milled in 1910 was: Au, 0.256 oz.; Ag, 33.568 oz.; CuO, 1.06, and PbO, 0.568 per cent.; Zn, present but not determined. A list of the minerals that have been recognized in the ore takes in the following: galena, zinc-blende, iron pyrite, chalcopyrite, bornite, argentite, stromeyerite, cerargyrite, hematite, pyrolusite, native gold, native silver, quartz, calcite, fluorspar, and alunite. Proustite is sometimes found, but its occurrence is rare. An aggregate which, to the naked eye, appears homogeneous, and has a silvery gray color, is frequently encountered in masses varying in size from several inches down to small grains, associated with pyrite, bornite, galena, and quartz. This was supposed to be tetrahedrite until a chemical analysis showed that there was no antimony present. The analysis gave:

	Per Cent.		Per Cent.
Ag,	18.5	As,	00.0
Au,	trace	Zn,	2.3
Cu,	35.7	Fe,	10.8
Pb,	7.3	S,	17.1
Bi,	00.0	Insoluble,	5.0
Sb,	00.0		96.7

Specific gravity, 5.65.

Specimens of this ore, submitted to Charles P. Berkey, of Columbia University, were pronounced by him, from microscopic evidence, to be aggregates. A polished surface of the ore under the microscope showed that it was an intergrowth of several simple metallic minerals which had developed a micrographic structure in solidification.

I found that in cyanide solution this particular ore acts very peculiarly. A sample ground so that all passed through a 200-mesh screen was treated by successive air-agitations in fresh

solutions of 0.94 per cent. of KCN, which carried also 0.5 per cent. of CaO and 0.05 per cent. of lead acetate. During the first 60 hr. of treatment, practically none of the silver went into solution, while the copper, on the other hand, was almost completely dissolved. At the end of 60 hr., solution of the silver commenced to take place, and was fairly rapid until the 100th hour. From 100 to 180 hr. the silver went into solution more slowly. At the end of 180 hr. the extraction of the silver was still only 87.86 per cent., while that of the copper was 97.73 per cent. The weight of potassium cyanide consumed or decomposed was actually greater than the weight of the sample. Naturally, the presence of any considerable quantity of this rich silver-copper compound would be very undesirable in the tailings when treated with cyanide solution, and, fortunately, it can be almost completely removed by careful table-concentration.

General Outline of the Milling Process.

The ore from the mine is hauled out by way of the No. 7, or lowest, tunnel in trains of 1-ton cars, by mules, and dumped into a 250-ton steel ore-bin at the top of the mill. An automatic feeder draws ore from the bin and drops it on a shaking-grizzly. The oversize from the grizzly falls into a gyratory breaker, and then is sorted on a picking-belt, where waste as well as high-grade ore is removed. This section of the plant has a capacity of 20 tons per hour, so that the required mill-tonnage may be crushed, sorted, weighed, sampled, and distributed to the two mills during the day shift. A complete flow-sheet of the milling- and cyanide-operations is given in Fig. 4.

Depending upon the distribution desired, from 50 to 56 per cent. of the crushed ore is sent to No. 1 mill, where it is crushed further by rolls, the sizing being done by trommels and hydraulic classifiers, and the concentrating by Harz jigs, Wilfley tables, and Frue vanners. The balance of the crushed ore from the breaker-plant is sent over a belt-conveyor to No. 2 mill, where the final crushing is done by twenty 1,200-lb. gravity stamps, the sizing by a Richards vortex type of hydraulic classifier, and the concentration by Wilfley sand-tables and Deister slimers. In the No. 2 mill the middlings from the Wilfley tables are reground in a 5-ft. grinding-pan, and join

the slime middlings from the Deisters to be re-treated on a Deister slime-table. The results obtained from the No. 2 mill, with its simple but thorough scheme of concentration, have been considerably better than those obtained in the stage-crushing mill during the same periods, and will be referred to in detail further on in this paper.

On account of the rapid decomposition of cyanide if solution is allowed to be in contact with the sulphides of the Tigre ore, the concentration in both mills is done in water, sufficient lime being added to the crushed ore at the breaker-plant to aid the settling of slime, and to make the tailings alkaline before coming in contact with solution.

The tailings from both mills flow by gravity to the tube-mill section of the plant, where they are classified in two drag-classifiers and the material coarser than 200-mesh dewatered in two 5- by 14-ft. tube-mills. The product of these tube-mills is elevated, and joins the product of three other tube-mills for classification in solution by three drag-classifiers similar to the first two. The sand from the last three classifiers goes to the corresponding tube-mills, and in this manner a closed grinding-circuit is made from which only material of the required degree of fineness may escape. Old tailings from the dumps at the rate of 75 tons per day are delivered to drag-classifiers Nos. 3 and 4 after being thoroughly screened, sampled, weighed, and mixed with lime and solution.

The slime from No. 1 and No. 2 classifiers is dewatered by means of two 24- by 12-ft. Dorr slime-thickening tanks followed by vacuum filtration. The slime from the other classifiers, which is in solution, is thickened in a 24- by 12-ft. Dorr thickening-tank, and joins the dewatered slime for continuous treatment in a series of eight 15- by 40-ft. Pachuca tanks. At this stage the pulp has a consistency of 2 of liquid to 1 of solid by weight, and sodium cyanide is added at the first treatment-tank to bring the strength of solution up to 0.35 per cent. in terms of KCN. Protective alkalinity is maintained at from 1.5 to 2.0 lb. of CaO per ton of solution. The use of lead acetate was discontinued after the first six weeks of operation. In flowing through the series of treatment-tanks the slime receives the equivalent of 48 hr. treatment. From the last tank it is allowed to discharge into two 24- by 12-ft.

BREAKER AND SAMPLING-PLANT.

1 250-ton steel Ore-Bin, 15 ft. by 30 ft.
1 Revolving-drum Ore-Feeder, 24 in. by 27 in.
1 Shaking-Grizzly, 30 in. by 78 in., 1.5 in. spacing.
1 No. 4 Style "K" Allis-Chal. Breaker set to 1.5 in.
1 Sorting-Belt, 2.8 ft. long, 32 in. wide, 6-ply, ⅛ in. rubber cover, speed 50 ft. per min.
1 Blake-D. automatic Weighing-Machine.
1 Belt-Conveyor, width 14 in., length 87 in., speed 160 ft. per min., 5-ply, ⅛ in. cover.
1 Revolving-Cone Sampler and Distributor, 5 per cent. cut, sending 80 tons to
1 Belt-Conveyor, width 12 in., length 179 ft., 5-ply, ⅛ in. cover, speed 200 ft. per min.
1 Sample-Crusher, 7 in. by 10 in.
1 Snyder Sampler, 27 in. diam., 10 per cent. cut.
1 Sturtevant Roll-Jaw Sample-Crusher, 2 in. by 6 in.
1 Set Sample-Riffles cutting 10 per cent. sample.
1 Shaking-Grizzly, 30 in. by 78 in., ¾ in. spacing.
1 No. 2 Austin Gyratory Crusher, set to ¾ in.
1 Broken-Ore Bin, capacity 120 tons.
2 Eccentric and Arc Fine-Ore Feeders.
2 Trommels, 4 ft. by 5 ft., ⅝ in. round holes.

OLD CONCENTRATING-PLANT.

2 Trommels, 4 ft. by 5 ft., ⅜ in. round holes.
1 Double-compartment, Bull-Jig, 4-mesh wire screen, 107 2⅝-in. strokes per min.
1 D. C. Bull-Jig, 4-mesh, 107 1⅝-in. strokes per min.
2 Trommels, 4 ft. by 5 ft., $\frac{3}{16}$ in. round holes.
1 Cole Dewatering-Wheel.
1 Pair Roughing-Rolls, 16 in. by 36 in., Cole pattern.
2 D. C. Jigs, 23½ in. by 35¾ in., 6-mesh, 170 ¾-in. strokes per min.
1 Bucket-Elevator 39 ft. cc., 18 in. belt, 15 in. buckets.
2 D. C. Jigs, 23½ in. by 35¾ in., 8-mesh, 170 ⅝-in. strokes per min.
2 Trommels, 4 ft. by 5 ft., 1½-mm. round holes.
1 Cole Dewatering-Wheel.
1 Pair 16 in. by 36 in. Rolls.
1 Baffle-Screen removing chips from water.
2 Richards-Coggin Classifiers.
1 Bucket-Elevator, 42 ft. cc., 18 in. 10-ply belt, speed 360 ft. per min., buckets 15 in. by 7 in. by 6 in.
1 5-ft. Spitzkasten with baffle.
1 Richards-Coggin 3-spigot Classifier.
1 Three-compartment Jig, 10-mesh, 170 ⅝-in. strokes per min., middlings from hutch.
1 Dewatering Feed- and Storage-Box.
2 5-ft. Huntington Mills, ¾-mm. screens.
5 No. 5 Wilfley Tables, one for re-treating Jig-middlings.
4 16 ft. Redwood Settling-Tanks.
1 Bucket-Elevator, 35 ft. cc., 12 in. belt, 9 in. buckets.
6 No. 5 Wilfley Tables.
1 No. 5 Wilfley Table re-treating middlings.
1 9 in. by 10 in. Aldrich Triplex Pump, 47 R. P. M.
2 V-Bottom Slime-Thickeners.
1 Frenier Spiral Sand-Pump, 10 in. by 54 in.
2 Callow Pulp-Thickening Cones, 8 ft. diam.
16 6-ft. Frue Vanners.

Figures denote short tons per 24 hr. Ratios denote consistency of pulp by weight of solids and weight of liquids.

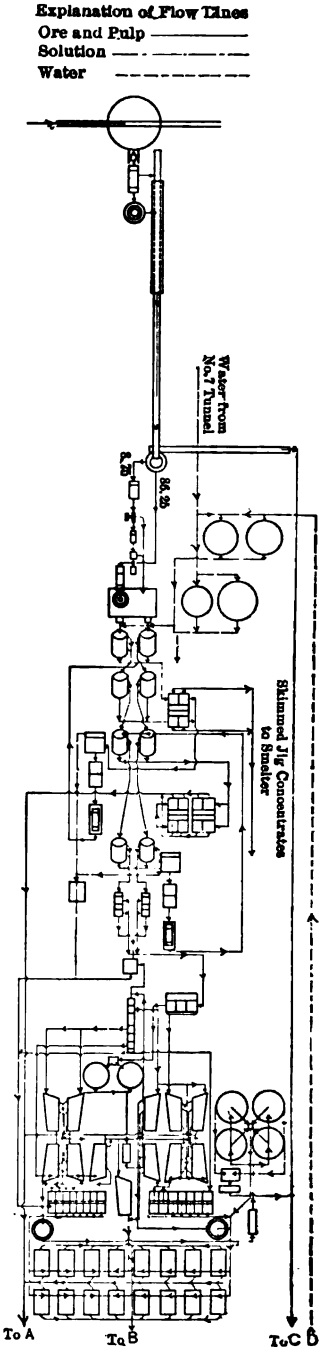
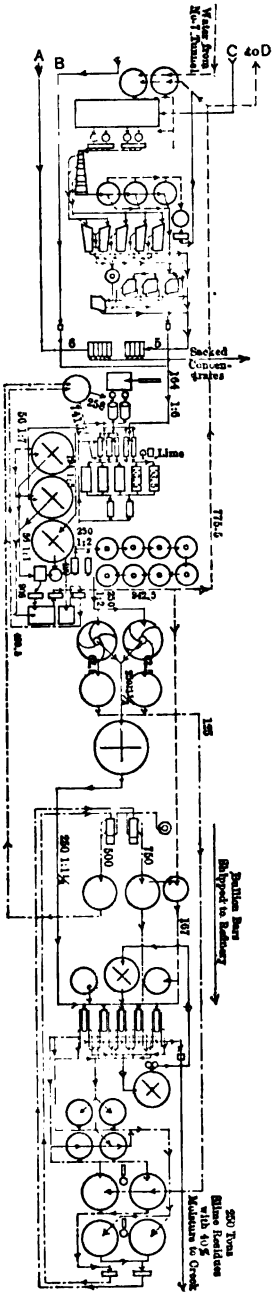


FIG. 4.—FLOW-SHEET FOR



NEW STAMP-MILL AND CONCENTRATOR.

2 16 ft. Redwood Water-Tanks.
1 350-Ton Broken-Ore Bin fed by 12 in. Belt-Conveyor and Traveling-Tripper.
4 Suspended Challenge Ore-Feeders.
20 1,200-lb. Gravity Stamps, 20-mesh screens, 4 in. chuck-block, 102 7½-in. drops per min.
1 Hydraulic Vortex-Type Classifer.
3 16 ft. Conical-Bottom Dewatering-Tanks.
1 9 ft. by 8 ft. Conical-Bottom Return-Water Tank.
1 9 in. by 10 in. Triplex Return-Water Pump.
4 No. 6 Wilfley Tables, 240 R. P. M.
1 No. 5 Wilfley Table, 240 R. P. M.
1 5-ft. Grinding-Pan for middlings.
3 Deister Slime-Tables.
1 Deister Slime-Table re-treating middlings.
2 Scobey Tailings-Samplers.
2 Concentrates-Bins.
1 15 ft. by 10 ft. Steel Solution-Tank.

CYANIDE-PLANT.

1 System Flexible Tailings-Conveyor Belts for collecting dump-tailings.
2 Suspended Tailings-Feeders.
2 4 ft. by 5 ft. Trommels for screening and mixing tailings with solution.
5 42 in. by 16 ft. Drag-Classifiers.
1 Lime-Bin and feeder, 1½ tons p. d.
5 5 ft. by 14 ft. Wet-Grinding Tube-Mills.
2 24 ft. by 12 ft. Dorr Dewatering-Tanks, Steel.
2 Bucket-Elevators, 19 ft. cc., 18 in. belt, 15 in. buckets, speed 400 ft. per min.
1 24 ft. by 12 ft. Dorr Slime-Thickening Tank, steel.
1 Wilfley Dewatering Vac. Filter.
1 Mechanical Mixer for Slime-cakes
2 Bucket-Elevators, 39 ft. 9 in. cc., 18 in. belt.
8 15 ft. by 40 ft. Tall Steel Tanks for Treatment by submerged Air-Agitation.
3 Pumps—vacuum, water and solution.
2 Water proofed Concrete Sumps.
2 24 ft. by 14 ft. Tigre-type steel Pulp-Thickening Tanks.
2 20 ft. by 10 in. Sand-Filter Tanks, steel.
1 Tigre-type bottom-drive mechanical agitation Slime-Storage Tank, 30 ft. by 12 ft., steel.
2 Merrill Precipitation-Presses, one 20-frame for strong solution, other 30-frame for weak solution.
1 No. 125 Crucible "Steele-Harvey" Tilting Oil-Furnace.
2 20 ft. by 10 ft. Steel Tanks for Barren Solution.
1 15 ft. by 10 ft. Steel Tank for Wash-Water.
1 20 ft. by 10 ft. Bottom-drive Mech. Agitating Tank for excess slime.
2 16 ft. by 10 ft. Redwood Conical-Bottom Water-Tanks—one for excess wash-water, other for cwt. mech'm.
5 5 ft. by 14 ft. "Parral"-Type Kelly Filter-Presses; each with 1,200 sq. ft. of Filter-area; spiral disintegrating-conveyor for breaking up and removing discharged cakes.
1 Scobey automatic Tailings-Sampler.
1 Air-Displacement Pump for returning excess slime.
1 20 ft. by 10 ft. Steel Agitating-Tank for excess slime.
4 15 ft. by 10 ft. Steel Tanks with sand-filter bottoms for clarifying solutions.
1 Zinc-Dust Feeder and Mixing-Cone.
2 20 ft. by 10 ft. Steel Tanks for strong pregnant solution.
1 Zinc-Dust Feeder and Mixing-Cone.
2 20 ft. by 10 ft. Steel Tanks for weak pregnant solution.
2 Aldrich 6 in. by 7 in. V. T. belt-driven Solution-Pumps.

CAPACITY.

Breaker and sampling-plant, 175 tons per 10 hr. Old concentrating-plant, 95 tons per 24 hr. New Stamp-mill and concentrator, 80 tons per 24 hr. Cyanide-plant, 250 tons per 24 hr.

Tigre slime-thickening tanks. Here the pulp is thickened to a consistency of 1.5 to 1, and then stored in a 30- by 12-ft. mechanically-agitated slime-storage tank of the bottom-drive type. Five Parral-type Kelly filter-presses draw the thickened slime from this storage-tank at a pressure of 32 lb. per sq. inch.

Two kinds of solution, known as strong and weak, are drawn from the Kelly filters. These solutions are of nearly the same cyanide strength, but the strong solution is obtained during the formation of cakes on the filter-leaves, while the weak solution is obtained in the washing-period. The strong solution, to which is added that which is obtained at the two Tigre slime-thickeners, is partly precipitated with zinc-dust, and then returned to the plant for treating tailings, as before. The weak solution forms a closed circuit, being precipitated completely with zinc-dust and then used again for washing the caked slime in the Kelly filters.

The precipitate, in spite of its high percentage in copper, is melted directly into bullion-bars in oil-fired tilting-furnaces. By a special method of fluxing and refining, which will be described later, the greater part of the copper and zinc in the precipitate is removed, and bars of from 800 to 850 fineness are obtained from precipitates of 60 per cent. average silver-content.

Bullion-bars and concentrates are sold, while the residues discharged from the filter-plant, which carry undissolved gold and silver to the value of from 80 cents to $1.10 per ton, are at present allowed to run to waste in the Tigre Cañon with from 42 to 45 per cent. of moisture. A dam is now being built below the plant to impound the residues.

Breaker-Plant and Sampling-Arrangements.

The run-of-mine ore is drawn from the 15- by 30-ft. steel ore-bin by a revolving-drum ore-feeder and falls on a 30- by 78-in. shaking-grizzly, the bars of which are spaced 1.5 in. apart and have a downward inclination of 14°. Owing to a device which gives a quick return-movement to the grizzly, the material on it is screened very thoroughly and made to run forward in a uniform stream to the gyratory breaker. Pieces of ore smaller than 1.5 in., which drop through the grizzly-bars,

are delivered from a spout on a 14-in. belt-conveyor that runs under and receives the discharge from the 32-in. sorting-belt. The sorting-belt takes the product of the No. 4 style K gyratory breaker. Three Chinese boys in 10 hr. are able to pick from the belt 1.5 tons of waste and 0.75 ton of high-grade ore. The average assays of waste and high-grade picked from the belt are as follows:

Waste—gold, trace; silver, 2.2 oz. per ton. High-grade—gold, 1.9; silver, 234.8 oz. per ton.

The percentage of recovery by sorting is as follows:

	Per Cent.
Of the gold in the run-of-mine ore,	3.7
Of the silver in the run-of-mine ore,	2.5
Of the value of gold and silver in ore,	2.7

The cost of sorting is about 4 cents per ton, of which labor is 3.2 cents, and power 0.7 cent.

All of the ore is weighed by means of a Blake-Dennison machine as it is carried on the 14-in. conveying-belt to the concentrating-mills. This conveyor discharges the ore into a machine which distributes it in the desired proportion between the two mills, at the same time cutting a 5 per cent. sample. The distributing-and-sampling device, Fig. 5, consists of a cast-iron cone with a single discharge-spout that revolves about a vertical axis over a stationary cone with three compartments. Between the two large compartments is an adjustable partition which may be set to various positions corresponding to percentages of the total ore that it is desired to give to the No. 1 mill. From the compartment corresponding to No. 2 mill a discharge-spout delivers the material to a 12-in. conveying-belt, which carries it to the stamp-battery ore-bin. From the small sample-compartment the portion of ore cut for sampling is dropped into a 7- by 10-in. Blake crusher, which reduces the size of the largest pieces to 0.75-in. ring. A Snyder sampler placed below the crusher takes out 10 per cent. of the original sample, and the large pieces in this sample are crushed in a 2- by 6-in. Sturtevant roll-jaw crusher, after which the amount of the sample is reduced again by sets of riffle-boxes until about 40 lb. is obtained for sending to the assay-office. The rejected portions of ore at each step in the sampling drop into the broken-ore storage-bin of No. 1 concentrator.

Concentration in No. 1 Mill.

A shaking-grizzly with bars spaced 0.75 in. apart is placed below the distributing-machine and screens the ore sent to No. 1 mill. The oversize from this grizzly is reduced to pass

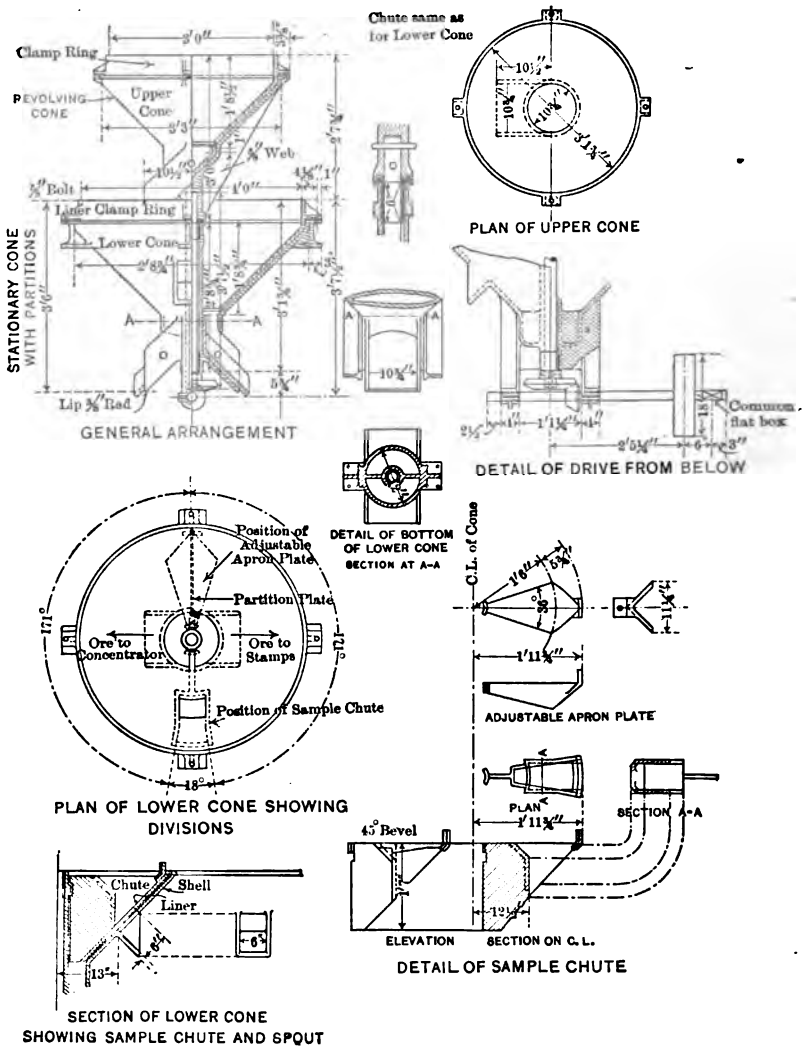


FIG. 5.—48-IN. DISTRIBUTING AND ORE-SAMPLING HOPPER.

a 1-in. ring by a No. 2 Austin gyratory crusher. The undersize from the grizzly and the product of the crusher are collected in a storage-bin of 120 tons capacity. Automatic feeders

draw the ore from this bin to the first trommels, where mill-water is added for the first time. The trommels are 4 ft. in diameter by 5 ft. in length, and have holes $\frac{5}{8}$ in. in diameter punched in plates $\frac{5}{16}$ in. thick. The oversize from the first pair of trommels is carried with water in a launder to a double-compartment bull-jig of the Harz type, which makes 107 $2\frac{3}{8}$-in. strokes per minute, and uses a 4-mesh wire screen with a mineral bed in which concentrates are collected for periodical removal by an attendant with a shovel. The tailings from the jig are crushed in a pair of 16- by 36-in. rolls, and are then elevated and screened in a second pair of 4- by 5-ft. trommels.

The second trommels have $\frac{3}{8}$-in. holes, and the product, consisting of pieces which range in size from $\frac{3}{8}$ to $\frac{5}{8}$ in., is delivered to a double-compartment bull-jig, similar to the first but having a $1\frac{3}{8}$-in. stroke. The tailings from this jig join those from the first for crushing, and return to the second pair of trommels.

The concentrates from the bull-jigs amount to about 10 per cent. by weight of the total concentrates of the mill.

The third pair of trommels, which takes the undersize from the second pair, has $\frac{3}{16}$-in. holes, and its product is sent to four double-compartment Harz jigs, which make 170 strokes per minute. Two of these jigs work on a 0.75-in. stroke, while the others use a $\frac{5}{8}$-in. stroke. The concentrates are drawn off continuously from iron hutch-doors, while the tailings are dewatered, crushed in a pair of 16- by 36-in. rolls, and then elevated to the third pair of trommels.

The fourth pair of trommels has 1.5-mm. holes and takes the undersize from the third pair. Its oversize is dropped to the 16- by 36-in. rolls, where it joins the tailings from the Harz jigs. Its undersize is sent to a pair of Richards-Coggin hydraulic classifiers, which sort out the coarser particles for concentration in a three-compartment Harz jig. This jig makes 170 $\frac{5}{8}$-in. strokes per minute and produces middlings that are further concentrated on a Wilfley table, while its tailings are sent to one of two 5-ft. Huntington mills for regrinding through 0.75-mm. punched screens.

The overflow from the first classifiers is delivered to a 5-ft. *spitzkasten*. The overflow of the latter, consisting of slime fine enough for vanner-treatment, is sent to V-bottom slime-thick-

eners. The spigot-product of the *spitzkasten* is divided into four classes by means of a three-compartment Richards-Coggin classifier. Three of the classes consist of sand particles, which are treated on Wilfley tables. The fourth class is the slime overflow, which is sent to the V-bottom tanks for thickening previous to treatment on Frue vanners.

There are 12 No. 5 Wilfley tables, one of which is used for the re-treatment of the middlings from the others. Eleven of the tables make four products: slime, which is drawn off nearest the head of the table and sent to the Callow pulp-thickening cones previous to vanner-treatment; sand tailings, which are sent to the cyanide-plant; middlings, which are re-treated on the twelfth Wilfley table; and concentrates, which are carried away in shaking-launders and delivered to a dewatering storage-bin.

There are sixteen 6-ft. Frue vanners. The feed for them is thickened to about 1 to 4, solid to liquid ratio. The concentrates are partly removed from the belts by rollers on the under side, which are wound spirally with strips of old rubber belting. Tailings from the vanners join the sand tailings from the Wilfley tables and are sent to the cyanide-plant.

TABLE I.—*Wear of Roll-Shells at No. 1 Mill.*

	Coarse Rolls, 16 by 36 in.	Fine Rolls, 16 by 36 in.
Material	Midvale steel.	Midvale steel.
Service	Material $\frac{3}{16}$-to 1.5-in. size crushed to 0.25-in. ring.	Undersize from $\frac{3}{16}$-in. trommels crushed to 2 mm.
Original weight of each shell	1,417 lb.	1,417 lb.
Average weight when discarded	230 lb.	301 lb.
Per cent. efficiency wear of steel	84 per cent.	79 per cent.
Number of days in service	143	119
Total tons of ore treated	14,163.48	11,630.37
Cost of steel per ton of ore treated	$0.0324	$0.0376

NOTE.—The shells are rough machined with both inside and outside faces parallel, and are fastened to the roll-centers by hard-pine wedges $\frac{11}{16}$ in. thick. Alternating with the wooden wedges are iron wedges $\frac{3}{8}$ in. thick and 6 in. long. Shells fastened in this manner seldom slip, but usually break when the steel is about 0.5 in. thick.

TABLE II.—*Wear of Trommel-Screens at No. 1 Mill.*

	$\frac{5}{16}$-in. Trommels.	$\frac{3}{8}$-in. Trommels.	$\frac{3}{16}$-in. Trommels.	1.5-mm. Trommels.
Thickness of plates used.....	$\frac{5}{16}$ in.	0.25 in.	No. 10 gauge	No. 16 gauge
Number of units in use.....	4	8	8	12
Life of screens, months.....	7	7	6	2
Tons of ore screened.....	21,000	21,000	18,850	5,850

In the Huntington mills, both punched and woven-wire screens have been tried, with results in favor of the former: 1-mm. No. 26 gauge punched-plate screens, although their screening-capacity was somewhat less, outlasted the Tyler 0.75-mm. Ton-cap wire screens, and cost only $0.35 per unit, as against $0.96 for Ton-cap screens.

TABLE III.—*Wear of Harz Jig-Screens at No. 1 Mill.*

	Bull-Jigs.	Middle Jigs. No. 1.	Middle Jigs. No. 2.	Sand-Jigs.
Mesh of screen used.....	4	6	8	10
Size of wire in screen.....	No. 12	No. 14	No. 16	No. 20
Diameter of openings, inch.....	0.145	0.082	0.060	
Life with steel wire, days.....	39			
Life with brass wire, days.....	59	120	120	40

NOTE.—Steel wire has been found to be more economical than brass wire for the screens of the bull-jigs, on account of the cost of the steel wire being only one-third that of the brass.

Concentration in No. 2 Mill.

The broken ore for No. 2 mill is carried from the distributor to a 350-ton ore-bin by a 12-in. conveying-belt, and is evenly piled in the bin behind the stamps by an automatic traveling-tripper. From here the ore is drawn and fed to the stamps by four Challenge ore-feeders of the suspended type.

The stamps are each 1,200 lb. in weight when shoes are new. The dimensions of the principal parts are as follows:

Shoe, ...	9 in. by 9 in., chrome forged steel.
Die, ...	9.25 in. by 8 in., chrome forged steel.
Head, ...	9 in. by 20 in., chrome steel.
Stem, ...	3.75 in. by 14 ft., extra-refined steel.
Tappet, ...	$9\frac{3}{8}$ in. by 15.75 in., chrome steel.

When running at full capacity the stamps make 102 7.5-in. drops per minute, and use a 4-in. steel-faced chuck-block. The screens are 20-mesh Ludlow-Saylor rolled slot No. 25 wire screens. Under such conditions, and with 6 tons of water supplied to the mortars for each ton of ore, each stamp has a capacity of between 5 and 6 tons per day.

The mortars weigh 10,000 lb. and are of the narrow single-discharge type, El Oro pattern, and were specially manufactured by the Allis-Chalmers Co. in two sections with a vertical dove-tail longitudinal joint, in order to facilitate transportation in wagons from the railway to the mine. The battery-posts are set in cast-iron shoes which hold them securely to the concrete foundations. Rubber sheets 0.25 in. thick are placed under both the mortars and cast-iron post shoes. The 20 stamps and the 12-in. belt-conveyor are driven by a 75-h-p. slip-ring induction-motor.

A typical screen-test on the pulp discharged from the stamp-battery is as follows:

	Per Cent.
On 20-mesh screen,	0.5
On 40-mesh screen,	15.4
On 60-mesh screen,	15.7
On 80-mesh screen,	8.4
On 100-mesh screen,	3.0
On 200-mesh screen,	9.2
Through 200-mesh screen,	47.8
	100.0

The pulp from the stamps is classified into two grades of sand and one grade of slime by a Richards vortex-type hydraulic classifier which has four spigots. The first two spigots deliver coarse sand for treatment on two No. 6 Wilfley tables. The last two spigots separate the greater part of the fine sand particles from the slime for treatment on three No. 6 Wilfley tables. The slime overflow from the classifier is thickened in three cone-bottomed tanks, each of which is 16 ft. in diameter. The slime, at a consistency of 1 part of solid to 4 parts of water, is drawn by siphon-discharge from the settling-tanks and sent to three Deister slime-tables, Fig. 6. (It should be mentioned here, to avoid confusion, that in this paper the term "slime" is used to designate a suspension of ore particles in a liquid, where the size of the particles is such that they will pass through

a standard 200-mesh woven-wire screen, irrespective of the physical character of the particles, which may be either crystalline or amorphous.)

The Wilfley tables, treating the coarse and fine sands from the hydraulic classifier, make three products—concentrates, middlings, and tailings. The middlings are reground in a Colorado Iron Works 5-ft. grinding-pan and join the middlings from the three Deister slime-tables for re-treatment on a fourth Deister table. The work done by the grinding-pan when working at a capacity of about 8 tons per day is shown by the following average screen-test:

Screen Size.	Feed. Per Cent.	Discharge. Per Cent.
On 40-mesh,	8.4	2.2
On 60-mesh,	13.4	10.0
On 80-mesh,	16.6	16.6
On 100-mesh,	12.0	15.0
On 200-mesh,	21.0	17.2
Through 200-mesh,	28.6	39.0
	100.0	100.0

It will be noted that there are no elevators in No. 2 mill, and that the scheme of treatment is very direct and simple. As a result, less power, fewer repairs, and smaller attendance are required for the treatment of the same tonnage of ore that is treated in No. 1 mill. It was not expected, however, that the No. 2 mill would give better recovery than the old mill from the Tigre ore; but, not only are the concentrates made by it of better grade than those from No. 1, but a better extraction is obtained, thus proving, for the case of this ore at least, that simplicity of treatment may aid recovery as well as reduce operating-cost.

TABLE IV.—*Screen-Tests Comparing Tailings and Concentrates from No. 1 and No. 2 Mills.*

Screen Size.	Tailings.		Table-Concentrates.	
	No. 1 Mill.	No. 2 Mill.	No. 1 Mill.	No. 2 Mill.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.
On 20-mesh	0.5	1.2	5.9	0.3
On 40-mesh	16.2	11.7	10.5	6.2
On 60-mesh	20.0	17.2	15.2	12.6
On 80-mesh	6.8	10.2	15.7	12.3
On 100-mesh	4.4	4.9	13.7	13.3
On 200-mesh	7.0	7.3	15.0	22.3
Through 200-mesh	45.1	47.5	24.0	33.0

The total average power, including lights, required for No. 1 mill, is 167.1 h-p., as compared with 86.2 h-p. for No. 2.

The costs per ton of ore milled are: No. 1 mill, $1.895, and No. 2 mill, $0.673 per ton.

The recoveries in the two mills, based on total value of the gold and silver produced in concentrates, compared with the value in the sorted mine-ore, are: No. 1 mill, 59.6, and No. 2 mill, 62.6 per cent.

TABLE V.—*Launder-Falls Used in No. 2 Mill.*

Location.	Size.		Dilution of Pulp.	Fall,
	Width.	Height.	S : L	Per Foot.
	Inch.	Inch.		Inch.
Stamps to classifier	8	8	1 : 6	$\frac{7}{8}$
Classified coarse sand to Wilfleys	6	6	1 : 3	$\frac{7}{8}$
Classified fine sand to Wilfleys	6	6	1 : 3	$\frac{3}{4}$
Slime to dewatering-tanks	8	8	1 : 20	$\frac{1}{8}$
Thickened slime to Deister tables	6	6	1 : 4	$\frac{3}{8}$
Sand tailings from Wilfley tables	8	8	1 : 5	1
Middlings from Wilfleys to pan	4	4	1 : 2	1
Concentrates from Wilfleys to bin	4	6	.....	$1\frac{3}{4}$
Slime tailings from Deister tables	6	6	1 : 7	$\frac{3}{8}$
Middlings from Deisters to 4th table	4	4	1 : 4	$\frac{5}{8}$
Concentrates from Deisters	4	6	.....	$1\frac{1}{8}$

Tube-Milling and Classification.

The equipment for regrinding the sand particles in the tailings consists of five 5- by 14-ft. wet-grinding tube-mills, made by the Power & Mining Machinery Co. These mills are equipped with spiral scoop-feeders, and are driven at the feed end by two sets of gears direct connected to individual motors by specially designed flexible leather-link couplings. The discharge ends are fitted with worm pebble-feed castings. Both ribbed liners and grating liners set on steel liner-plates are in use, with results in favor of the latter. The mills operate to best advantage when filled with pebbles to about 6 in. above the center-line. In spite of high first-cost, it has been found more economical to use only the best grade of imported flint pebbles, ranging in size from 2 to 4 in. in diameter. Quartz rock, obtained locally, was unsatisfactory, not only in the amount needed to keep the mills loaded, but also in efficiency of grinding and in wear on the linings. The sand fed to the tube-mills

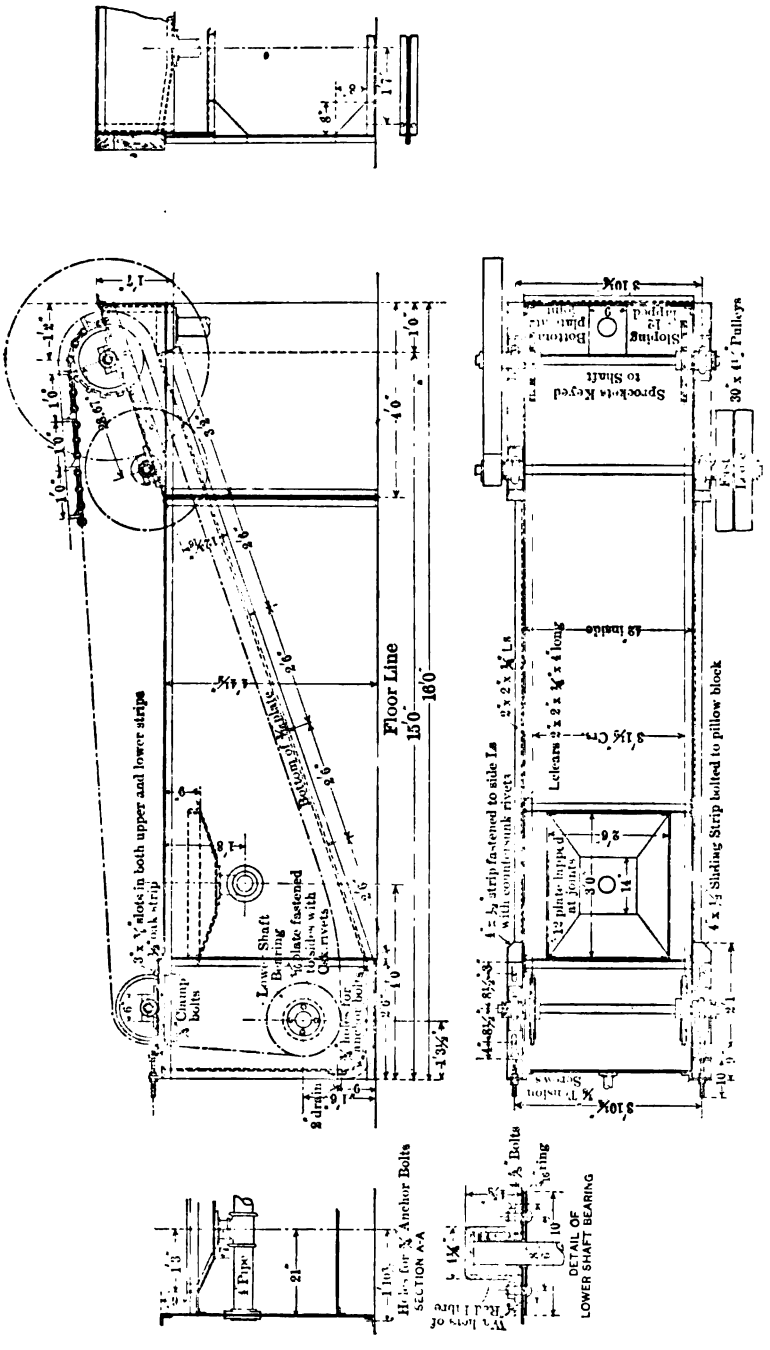


FIG. 7.—DETAIL OF 42-IN. DRAG-CLASSIFIER.

is maintained at a dilution of a little less than 1 part of solution to 1 part of sand by weight. Feed thicker than this deadens the blows of the pebbles in the mills and lowers the grinding-capacity, while a more dilute feed carries the sand through the mill before the particles have had sufficient grinding done upon them. The power consumed by each tube-mill ranges from 43 to 48 h-p., according to the height of the pebbles in the mill. While starting, each mill requires from 110 to 120 h-p. for a few seconds until the mill is brought up to its normal speed.

In order that the tube-mill plant may be capable of grinding the sand from 170 tons of concentrator-tailings, along with 75 tons of sand tailings from the dump, it is necessary to insure that the size of the sand particles from the concentrating-mills does not exceed 0.75 mm. When sand coarser than this limit is sent to the tube-mills, the capacity in product finer than 200-mesh diminishes very rapidly. Under normal conditions the slime delivered to the treatment-tanks has from 80 to 85 per cent. of material finer than 200-mesh and less than 5 per cent. coarser than 100-mesh.

The drag-classifiers (Fig. 7) which make the separation of sand from slime are in reality flight-conveyors, which elevate and dewater the particles of tailings that settle in steel boxes of special shape. It will be noticed from Fig. 7 that the overflow for the slime is such that the surface-velocity in the classifier is reduced to a minimum and so facilitates the settling of fine sand.

TABLE VI.—*Screen-Tests on Classifiers Nos.* 1 *and* 2, *Pulp-Dilution at* 1 : 6.

	On 100-mesh.	On 150-mesh.	On 200-mesh.	Through. 200-mesh.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Feed	56.0	3.2	2.1	38.7
Slime product	.....	.....	9.5	90.5
Sand product	91.1	3.7	2.2	3.0

In connection with the tube-mill plant is a system of 10-in. flexible belt-conveyors for transporting tailings from the old dumps below the mill to a storage-bin, where they are thoroughly screened, mixed with lime, automatically sampled and weighed,

and then made into a pulp with 0.15 per cent. of cyanide solution. The construction of these conveyors is such that the sand beneath them at any place may be excavated while the belt is running, and the framework sinks gradually into the new position. Loading-hoppers may be bolted to the framework at any point along the belt, and the tailings shoveled from under and from both sides of the conveyor. The operation of lowering usually proceeds from the tail end of the belt forward, and the trenches are made from 3 to 5 ft. deep by 12 ft. wide at each side. The head end of the first conveyor is driven by means of line-shafting and a universal coupling at the top of the storage-bin. When it is desired to change the alignment of this conveyor, its length is shortened to about 100 ft., and, as the work of excavation proceeds, new sections are added until the full length of 500 ft. is reached. Such a change is required only at intervals of six months or longer. At the present time only one other conveyor is used in connection with the first. This second conveyor, of a maximum length of 450 ft., is driven by a 5-h-p. motor, and discharges at any desired angle on the first. Eventually a third conveyor will be required to enable the complete removal of all the tailings in the dumps. Four Chinese shovelers load from 60 to 80 tons per 10 hr. on the belts, and they usually have two loading-hoppers in use at one time. A traveling-cableway with grab-buckets was planned at first for the excavation of the old dumps, but, after making a careful analysis of first-cost and operating-expense, it was concluded that a cableway installation would not be advantageous on the work of handling such a small tonnage of material per day. The cost of conveying and screening with the flexible conveyor arrangement is $0.11 per ton.

Dewatering.

The removal of mill-water from the tailings previous to treatment in solution is an obvious necessity in a cyanide-plant treating such rich silver-ore as that of El Tigre, for every ton of water introduced to the cyanide-plant has to displace a ton of weak solution somewhere, and even the weak solution here is stronger than the working-solution at most mills treating gold- or low-grade silver-ores. To avoid excessive waste of cyanide the dewatering of the tailings at El Tigre has to be very thorough.

No difficulty was experienced in the removal of water from the sand portion of the tailings, for, in the elevation of this material by the drag-classifiers, the sand is drained to from 20 to 25 per cent. of moisture by the time it reaches the discharge end of the machine, where solution sprays dilute it and wash it into the tube-mills.

In the case of the natural slime separated from the sand in the first two classifiers, a surprising amount of difficulty was encountered while trying to reduce the moisture-content to 40 per cent. It will be seen from the flow-sheet that two Dorr slime-thickening tanks, 24 ft. in diameter by 12 ft. in depth, were provided for the part dewatering of this slime. The rate of settlement was found to be very slow, however, owing to the amorphous character and extreme fineness of the material. The feed to the tanks consisted of a pulp which was normally at a dilution of 1 to 17, and, even when allowing the thickened slime to discharge at 1 to 4, clear water overflows were not obtained until accelerating-devices similar to those described by B. L. Gardner, which were used at the Sons of Gwalia mine, Australia, were placed in the tanks, and lime added to the ore at the breaker-plant in the proportion of 2 lb. of lime per ton of ore.

The final dewatering of the thickened slime is done in vacuum filters of local design, in which the cakes are thrown off and made into a pulp with solution of 0.15 per cent. of cyanide strength.

A curious characteristic of the natural slime in the ore at El Tigre is that, *per se*, it does not settle to a thicker pulp than 1 : 3 in 48 hr., even with the aid of lime or alum dissolved in the water. Lime, alum, cyanide, and similar salts have a marked effect in accelerating the settlement of the slime down to a point at about 1 : 6 dilution, after which further settlement is extremely slow, and finally reaches a point at between 1 : 3 and 1 : 4, where the slime seems to have reached what may be termed its critical condition, beyond which settlement is scarcely perceptible, even in periods of 24 hr. This critical condition of the slime is practically the same whether it has settled in clear water or in a solution of some salt.

Treatment.

The cyanide-treatment of the tailings really commences in the tube-mills, where the material is in contact with a solution of 0.15 per cent. of KCN. By the time that the slime has reached the first treatment-tank about 65 per cent. of the gold and about 50 per cent. of the silver in the tailings have been taken up by solution.

The treatment-tanks are of the Pachuca type,[1] 15 ft. in diameter by 40 ft. in height, and are arranged in a series of eight for continuous treatment. The total capacity of the eight tanks, with a hydraulic grade of 4 ft. between the first and last, is 560 tons of dry slime in a pulp, with dilution of 1 to 2; so that, when treating tailings at the rate of 250 tons per day, it is safe to say that all material in the tanks obtains at least 48 hr. of treatment. The chance of any material escaping from the series with less than two days' treatment is extremely small, owing to the number of tanks in the series; while the gain in time by not having to fill and discharge the tanks individually, coupled with the saving of corresponding expense in attendance, far more than offsets any real or supposed advantage that there might be in more positive control of the treatment when tanks are charged separately.

The slime in entering the first tank of the series falls on sodium cyanide cake in a perforated iron basket. In this way the solution is brought up to 0.25 per cent. in terms of KCN.

Air for agitation is compressed to 25 lb. per sq. in., and, to maintain a vigorous movement of slime in the tanks, from 80 to 100 cu. ft. of free air per minute per tank is required. The central air-lift pipe of each tank is 18 in. in diameter, and only extends to a point at two-thirds of the height of the tank, thus producing a submerged form of agitation, in which the air comes in contact with more particles of slime than in the usual form, in which the central pipe extends above the level of the slime.

For use in cases of emergency, high-pressure air-connections are supplied, by means of which air at 80 lb. pressure may be used inside the central tube, or on the outside by the aid of a temporary hose and air-pipe. As a further precaution, a cen-

[1] *Trans*, xlii., 807 (1911).

trifugal pump is placed under the tanks, and may draw off the contents of any tank in the series. For the purpose of dislodging any sand which might collect in the pipe-connections between the tanks, a high-pressure air-connection is furnished, which, in conjunction with the three-way iron cock at each of the connections, enables the operator to clean out the pipes whenever necessary.

The continuous-treatment system, as employed at El Tigre, has proved very satisfactory, and requires no attention except the occasional adjustment of valves by the shift-boss.

Slime-Thickening and Storage.

The normal dilution of the slime in the treatment-tanks is 1 to 2. For economical filtration, this pulp is too thin, and accordingly two 24- by 12-ft. slime-thickening tanks are used. These tanks give an almost perfectly clear solution overflow and discharge from the bottom a slime of about 1 to 1.5 dilution. Instead of using the Dorr type of continuous thickening mechanism for these tanks, spiral blades of sheet-steel supported from a central vertical shaft are used. This style of slime-thickening tank is illustrated in Fig. 8. The worm-gear wheel which drives the mechanism is keyed to the central shaft on a feather key, and permits the blades to be raised or lowered as desired. It will be noted from the illustration that the design of the blades is such that thickened slime anywhere on the bottom of the tank is continuously drawn in to the central discharge.

The thickened slime is stored in a 30- by 12-ft. tank, in which it is agitated by two pairs of steel arms with hinged iron paddles that are driven from beneath the tank, as shown in Fig. 9. The collar of this tank is 70 ft. above the center-line of the Kelly filters, and a mean gravity pressure for filtration of 32 lb. per sq. in. may be obtained at the filters while cakes are forming simultaneously in all five units.

Clarification of Solutions.

For economical and satisfactory precipitation of the gold and silver from cyanide solution it is an essential that the solution be perfectly clear, that is, that there be no solid matter in suspension previous to the addition of the zinc-dust. The

overflowing solution from the thickening-tanks just described, while apparently clear to a casual observer, in reality still has a certain amount of very fine clay and silica particles in suspension. In order to clarify this and also the cloudy solutions

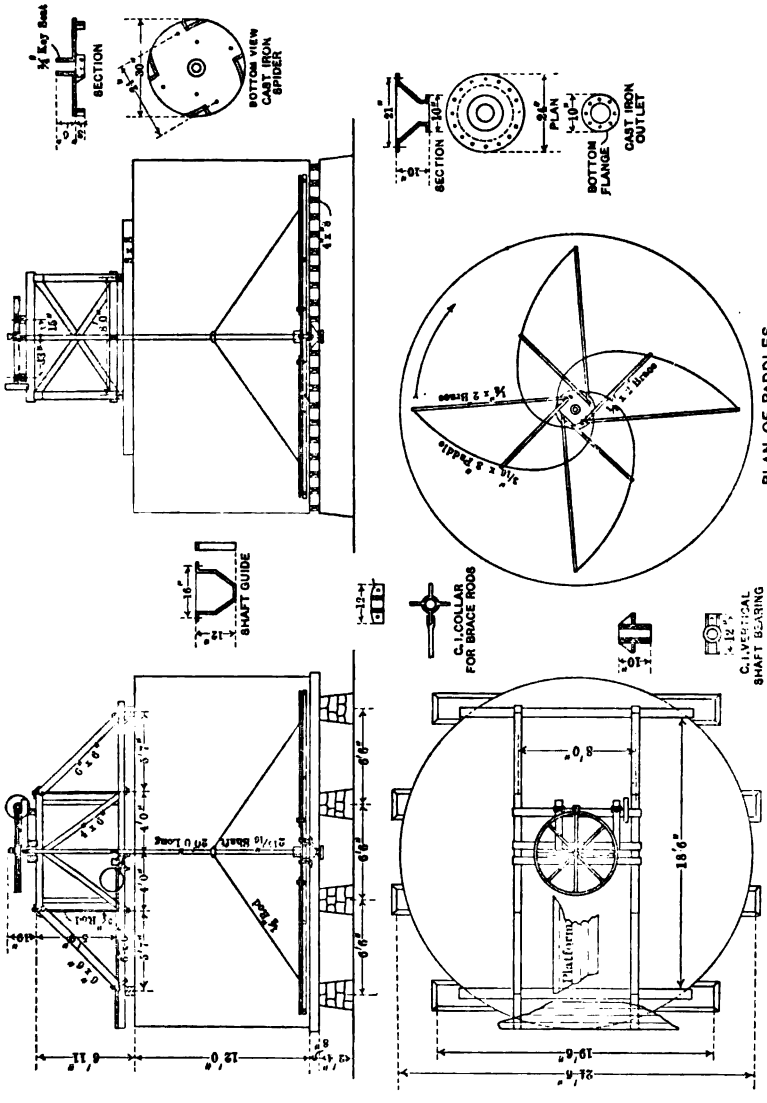


FIG. 8.—DETAILS OF TIGRE-TYPE THICKENING-TANK.

obtained at the filter-plant, sand-filter tanks of the usual type were tried at first, but were afterwards altered, and the sand-filter bottoms replaced by locally designed vertical leaves. Each leaf is 42 by 120 in. in size, and is made of grooved wooden

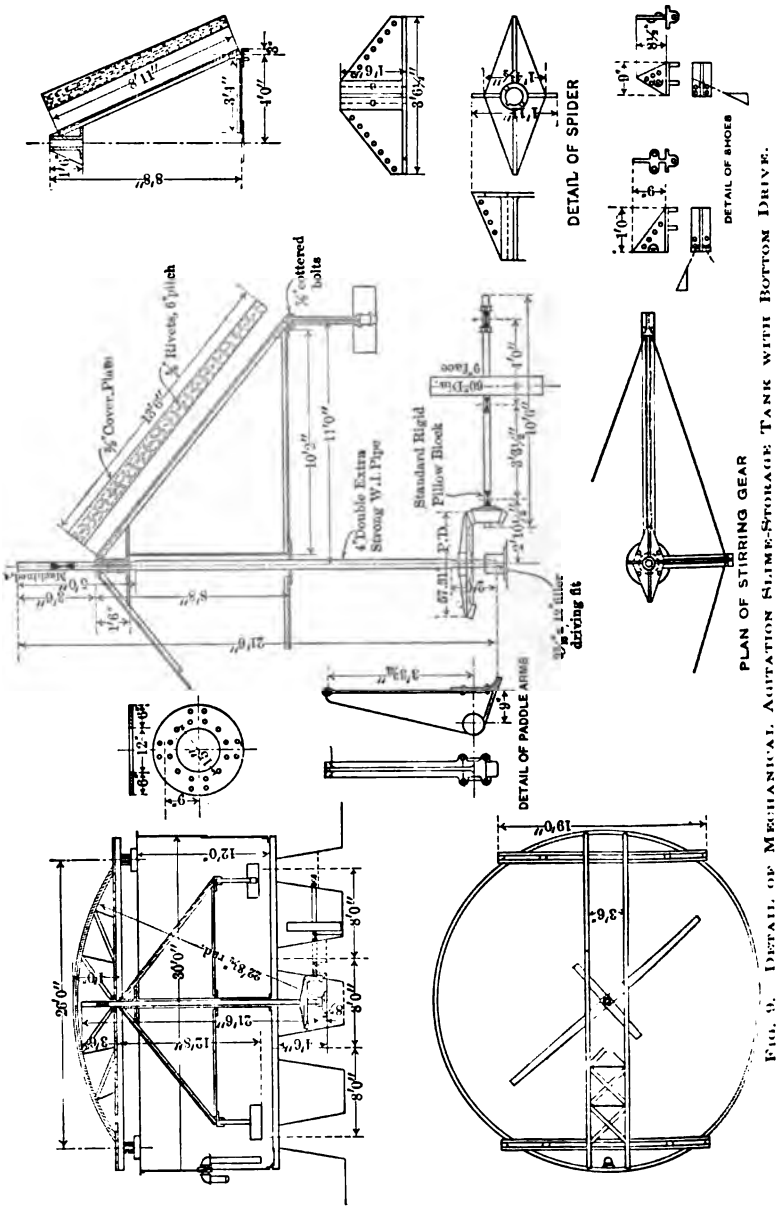


FIG. 9. DETAIL OF MECHANICAL AGITATION SLIME-STORAGE TANK WITH BOTTOM DRIVE.

slats placed above a perforated iron pipe and inclosed between two sheets of canvas that are sealed all around by P. & B. paint and a bolted iron frame. In this way a perfectly tight filter-leaf is obtained which has no stitching or nail-holes exposed at any part of the surface. The leaves are supported on channel-irons with their under sides 12 in. above the bottom of the tank, thus permitting the greater part of the sediment to accumulate on the tank floor. The leaves are spaced 18 in. apart so that, once in six weeks, the cake which has accumulated may be removed by hand with wooden scrapers, and the canvas scrubbed with a weak solution of hydrochloric acid. The two upper clarifying-tanks are 20 ft. in diameter by 10 ft. high, and the sediment on the bottoms may be washed out by a hose into the 30- by 12-ft. storage-tank. At the filter-plant there are four 15- by 10-ft. clarifying-tanks, which are operated in a similar manner except that, when it is desired to clean them, the solution is drained off thoroughly, a wash of water given, and the sediment then allowed to run to waste.

Filtration.

The choice of the Kelly type of slime-filters (a photographic view of a Kelly filter is given in the paper of W. P. Lass on the cyanide-plant at the Treadwell mine, Alaska)[2] for the Tigre cyanide-plant depended not so much upon maximum efficiency in filtration as upon conservation of water. Owing to the location of the plant in a section of the country which is periodically subjected to prolonged dry seasons, and to the fact that the supply of mill-water is derived solely from the mine and varies according to the rainfall at different seasons, it was imperative to insure economy of water in designing the plant. Only after a careful survey of and numerous tests with various types of filters which would operate successfully on the clay slimes of El Tigre, it was decided to install five units of the Parral type of Kelly presses. After solving certain initial mechanical difficulties inherent in filters of this sort, the Kelly plant has proved fairly satisfactory and gives excellent metallurgical results, though at high operating-cost.

The filter consists of a number of vertical leaves attached to

[2] *Trans.*, xlii., 803 (1911).

a movable carriage that runs in and out of an inclined sheet-steel cylinder. The movable head of the cylinder is made to seal the lower end hermetically when the carriage is run in for the filtering-operation. When the carriage is run out with the cages on the leaves, clear water and compressed air are forced from the inside of the leaves outward, and throw off the cakes into a large steel-lined trough, at the bottom of which a spiral ribbon-conveyor breaks up the cakes of filtered residue and pushes it forward to the discharge-launder. The complete removal of all slime that may adhere to the leaves, after discharging with water and air from the inside, is effected by directing a small stream of water at high pressure through a hose between the leaves.

The slime-cake, after air-drying on the leaves, contains only from 24 to 28 per cent. of moisture, but, at the present time, water is added during the discharge until there is about 45 per cent., at which point the slime may be conveniently carried off in a launder.

Each filter has 13 leaves, composed of an iron-pipe frame, with sewed canvas, inclosing vertical grooved wooden slats. The total filter-area of each press is approximately 1,200 sq. ft., and the capacity 50 tons of dry slime per day. The normal cycle of filtering-operations is:

	Time. Minutes.	Pressure. Pounds.
Filling with slime,	4	
Forming cake on the leaves,	20	10–32
Discharging excess slime and filling with solution,	6	4
Washing the cakes with barren solution,	30	25–34
Discharging excess solution and filling with water,	6	4
Washing with water,	4	25
Blowing back excess water and drying cakes with air,	5	10–15
Discharging cakes and returning carriage,	40–60	

Total time of cycle, from 1 hr. 55 min. to 2 hr. 15 min.

In order to prevent a building-up of dissolved gold and silver in the excess solution used for washing, this solution is run into the filters instead of slime once every shift, and in this way the efficiency of the cyanide wash is made dependent only upon the skill of the operator and the completeness of the precipitation of the weak solution.

The excess slime is discharged by gravity from the presses to a tank, from which it is returned to the filling slime-storage tank by two Frenier sand-pumps in series. Excess wash-solution and water are returned to their respective filling-tanks by compressed air admitted to the filter-shells.

All five presses are operated as one unit from a single point of control, where the operator has the valve-levers conveniently arranged.

Once in each month a wash of dilute hydrochloric acid is given the leaves by cutting out one press from the rest and pumping acid through the leaves for about 30 min. The strength of the acid is determined by testing the filtrate. Commercial hydrochloric acid is added to the acid-storage tank until the filtrate gives a distinct red coloration to the litmus test-paper.

During the first 3 min. of filtering or washing, the effluent solutions are usually cloudy, and, at such times, are sent to clarifying-tanks before being precipitated with zinc-dust.

Precipitation.

The Merrill process of zinc-dust precipitation is used. The zinc-dust comes in zinc-lined cases, and great care is taken to prevent oxidation. The feeders are of the improved hopper-and-worm type, which delivers a measured quantity of dust continuously to a small multiplex tube-mill, in which the dust is thoroughly emulsified with a stream of pregnant solution which comes from the upper clarifying-tanks. The emulsion flows from the little tube-mill continuously through a rubber hose placed in a vertical pipe connected with the suction of a 6- by 7-in. vertical triplex belt-driven Aldrich pump, which elevates the solution to the Merrill precipitation-press placed in the melting-room. The use of the little tube-mill for emulsifying the zinc-dust, when precipitating silver solutions, insures a uniform feed of zinc-dust, brightens and subdivides the particles of zinc, and prevents trouble with the pumps from formation of zinc-cake on the valve-seats.

Strong solution is precipitated at the rate of 500 tons per day in a triangular 52-in. 20-frame press, the frames being 3 in. deep. Weak solution is precipitated at the rate of from 600

to 750 tons per day in a similar press that has 30 frames. Since the strong solution is returned for the treatment of fresh tailings, complete precipitation is not required, and so a considerable saving in zinc-dust may be obtained. The weak solution, however, which is used for washing in the filter-plant, is precipitated as completely as practicable. The strong-solution press is cleaned up every week, while the weak-solution press is usually run for two weeks. Both presses have the discharge-cocks at the top in order to maintain a constant level of solution. The concrete floor under the presses drains to a small sump, from which all floor-washings are picked up and returned to the presses by means of a small auxiliary pump and motor, which are in use only about 5 min. each day. For clean-up a large sheet-steel tray on wheels is run under the press, and the precipitate, after being partly dried with air in the press, is scraped out of the frames and allowed to fall into the tray.

The strong solution heads average 0.02 oz. of gold and 4.50 oz. of silver per ton. A typical analysis of the precipitate from the strong solution is: Au, 0.31; Ag, 61.96; Cu, 8.20; CuO, 3.01; Zn, 1.70; ZnO, 5.85; SiO_2, 4.80; Al_2O_3, 4.50; Fe_2O_3, 1.50; CaO, 2.12; undetermined, 6.05; total, 100.00 per cent.

The heads of the weak solution average 0.016 oz. of gold and 3.2 oz. of silver per ton. The barren weak solution averages 0.001 oz. of gold and 0.16 oz. of silver for the entire period between clean-ups, being higher at the beginning and lower at the end of the time. A typical analysis of the precipitate from the weak solution is: Au, 0.20; Ag, 41.82; Cu, 19.80; CuO, 4.01; Zn, 0.90; ZnO, 10.00; SiO_2, 4.30; Al_2O_3, 3.70; Fe_2O_3, 1.30; CaO, 2.16; undetermined, 11.81; total, 100.00 per cent.

The zinc-dust used for precipitation contains 94.7 per cent. total zinc, of which 92.7 per cent. is in the metallic state.

Melting and Refining.

The precipitate taken from the presses is dried in an electrically-heated car, which has a series of superimposed shallow pans, under which the resistance-coils are placed. On account of the large amount of moisture given off from the precipitate while drying, the resistance-wires have to be protected by an asbestos-paint covering in order to prevent corrosion. The car has a capacity for drying 1,000 lb. of precipitate in 10 hr. and takes 4.5 kw. of power.

Melting is done in two No. 125 crucible Steele-Harvey oil-burning furnaces of the tilting type. The object of the melting is to fuse the precious metals and to remove as much of the copper and zinc as possible in a slag which will also carry off the baser impurities, such as alumina, silica, lime, and iron. To effect this purpose, fluxes are added to the precipitate in such proportions that the oxides of copper and zinc find sufficient finely-powdered silica present to unite with it and form subsilicates; the oxides of iron, aluminum, and calcium find sufficient borax and silica to dissolve or unite with them; while a small amount of sodium bicarbonate and a little fluorspar are added to make the slag fluid.

Flux for Strong-Solution Precipitate.

Analysis of Precipitate.	Silica in Parts Per 100 to be Added.	Borax in Parts Per 100 to be Added.	Fluorspar.	Soda Bicarbonate.
Per Cent.				
Au..... 0.30				
Ag.....63.13				
Cu..... 1.5				
CuO..... 3.14	Plus, 2.4	(Factor, 0.76)		
Zn..... 2.0				
ZnO..... 3.23	Plus, 2.4	(Factor, 0.74)		9.1
SiO_2.....11.6	Minus, 11.6			SiO_2 factor, 1.76
Al_2O_3..... 5.7	Plus, 10.0			Soda factor, 1.60
Fe_2O_3..... 1.3	Plus, 1.5	5.0	0.6 Borax factor, 3.79 SiO_2 factor, 1.14	Fluorspar, 0.1
	5.7	5.0	0.6	9.1

Flux 100 parts of precipitate with 5.7 parts silica, 5 parts borax, 0.6 part fluorspar, and 9.1 parts soda bicarbonate.

When fluxed in this manner the precipitate melts down very nicely. As the charge sinks in the crucible, fresh precipitate, mixed with flux, is added about every 20 min. until the crucible is nearly full of metal and slag in a state of quiet fusion. The slag is then poured into a conical pot, in which a crust is allowed to form to a thickness of 1 in., the molten slag inside the crust being then allowed to run out on cast-iron floor-plates. The crusts from this operation usually contain a small button of gold and silver at the point of the pot, and this button is broken off, and either put back in the crucible, or, if matte be present, is saved for a separate treatment with other similar buttons. The crusts themselves carry sufficient gold and silver to warrant a remelting, while the molten slag that is run on the floor is low in value, and, once a month, is weighed, sampled, and returned to the head of the mill.

After two to three pourings of slag, the molten metals in the crucible are subjected to a refining process, which depends upon the rapid formation of the oxides of zinc and copper when these metals are exposed to the air at high temperature in the molten state. With the furnace tilted forward, the operator throws a handful of bone-ash over the surface of the molten metal. This thickens whatever slag still remains, and enables it to be easily removed by means of spirals of 0.5-in. round iron. The air striking the exposed surface of the metals rapidly oxidizes the zinc and copper. After a couple of minutes, borax is thrown on and dissolves the oxides, forming a slag, which is thickened with bone-ash and removed as before. After about two repetitions of this refining, the precious metals are ready for pouring. The furnace is tilted back and the heat applied for about 5 min. with the cover over the crucible. A step-bottomed truck, with bullion-molds that have been heated and greased with a mixture of oil and flake graphite, is then run in front of the furnace, the mold on the highest step being set in front of the spout. The bars of bullion are poured by tilting the furnace forward over each of the molds in turn. After removal from the molds and cooling, the bars, which weigh from 1,200 to 1,300 oz., are chipped, sampled by drilling top and bottom, and stamped, ready for shipment. By this process, in spite of the high content in copper carried by the precipitate, bars from 850 to 900 fine are obtained.

From 20 to 30 lb. of flue-dust is collected and removed each month from a large chamber outside the melting-room through which the furnace-gases are led. A typical analysis of the flue-dust is: Au, 0.01; Ag, 21.39; Cu, 10.40; Zn, 7.50; SiO_2, 11.08; Al_2O_3, 2.90; Fe_2O_3, 2.30; undetermined, 44.42 (chiefly particles of borax, fluorspar, etc.); total, 100.00 per cent.

For the remelting of slag-crusts, scraps of iron box-bands are added to the slag in the proportion of about 0.5 per cent. of the weight of the slag, and the slag put through the furnace until a button of sufficient size to collect in the mold is obtained. Slag assaying Au, 1.98; Ag, 650 oz. per ton; Cu, 7.5; Zn, 13.5; SiO_2, 28.1; Al_2O_3, 11.34; Fe_2O_3, 4.76 per cent. was reduced in this manner to, Au, 0.06 and Ag, 32.34 oz. per ton; yielding metal that assayed Au, 0.147; Ag, 40.183; and Cu, 42.4 per cent.

Construction-Costs.

The building of the cyanide-plant, the No. 2 mill, and the remodeling of No. 1 mill, were done under somewhat adverse conditions, owing to the revolution in Mexico at the time, and consequent unsatisfactory labor-conditions, but the work was completed in 14 months from the time that the preliminary plans and estimates were approved by the directors of the company.

The following unit-cost figures on the construction-work may be of interest as an example for a property situated in Northern Mexico off the railroad:

Excavation, medium-hard ground with boulders, 16,103 cu. yd. done at an average cost of $0.64 per cu. yd.

Cement masonry, with stone partly dressed, 1,850 cu. yd. done at an average cost of $7.40 per cu. yd.

Cement concrete, 10 per cent. of which was reinforced, 972 cu. yd. done at an average cost of $12.28 per cu. yd.

Carpentry work, buildings, launders and miscellaneous, 320.3 M. board feet of lumber, put in at an average cost (including cost of the lumber) of $84.24 per M.

Steel tanks, assembling, riveting and calking, 247 tons of steel were put up at an average cost of $26.38 per ton.

The total cost of the cyanide-plant was $221,207.79, of which $11,203.39 was spent on testing and preliminary investigations.

The total cost of the No. 2 mill was $50,103.87, made up as follows:

Excavation,	$2,517.68	Supervision,	$1,479.19
Foundations,	9,812.32	Electrical work,	573.23
Carpentry,	14,723.24	Painting,	230.89
Local handling,	761.32	Miscellaneous,	245.86
Machine-work,	2,710.04	First-cost, freight, and duty,	16,990.97
Testing,	59.13		
			$50,103.87

Operating-Costs.

TABLE VII.—*Unit-Costs of Cyanide-Plant Supplies at El Tigre.*

Mine quartz for tube-mills, per lb.,	$0.00379
French flint pebbles, size A, per lb.,	0.02138
Danish flint pebbles, sizes C and D, per lb.,	0.02397
Zinc-dust, 92 per cent. metallic zinc, per lb.,	0.075
Sodium cyanide, 130 per cent. KCN, per lb.,	0.2247
Burned lime, 38 to 40 per cent. CaO, per lb.,	0.0155
Borax glass, per lb.,	0.1375
Sodium bicarbonate, per lb.,	0.07
Fluorspar, per lb.,	0.0745
Sodium nitrate, per lb.,	0.085
Graphite crucibles, No. 125, each,	9.54
Gas-oil for melting-furnace, per gal.,	0.375
Gasolene, per gal.,	0.4385
Hydrochloric acid, commercial, per lb.,	0.119
Lead acetate, per lb.,	0.12

TABLE VIII.—*Consumption of Cyanide-Plant Supplies.*

Sodium cyanide,	5.08 lb. per ton of tailings treated.
Lime, 52 per cent. CaO,	23.0 lb. per ton of tailings treated.
Zinc-dust,	1.3 lb. per ton of tailings treated.
Ratio Zn to Au plus Ag as 1.52 to 1.	
Gas-oil for melting,	0.025 gal. per oz. of bullion produced.
Flint pebbles,	8.7 lb. per ton of tailings treated.

TABLE IX.—*Milling-Costs Per Ton of Ore Milled.*

	November, 1911.		October, 1911.	
	No. 1 Mill.	No. 2 Mill.	No. 1 Mill.	No. 2 Mill.
Crushing	$0.496	$0.413	$0.566	$0.405
Regrinding	0.269	0.021	0.291	0.029
Concentrating	0.961	0.239	1.225	0.521
Construction	0.169	.....	0.189	.....
	$1.895	$0.673	$2.271	$0.955

The cost of cyanide-treatment at El Tigre is very high, owing chiefly to the large consumption of cyanide. The following are representative average costs for the various steps in the process:

Conveying and screening tailings,	$0.11
Tube-milling,	0.65
Classification, elevation, and dewatering,	0.22
Treatment,	1.50
Filtration,	0.30
Precipitation,	0.14
Melting,	0.19
Total,	$3.11 per ton.

Recovery.

From 62 to 64 per cent. of the gold and silver in the mine-ore is recovered in the form of concentrates and sorted high-grade. The recovery by cyanide-treatment of the concentrator-tailings is about 82 per cent., and of the dump-tailings about 85 per cent. The total recovery by sorting, milling, and cyaniding is from 93 to 95 per cent. Owing to the short time that the plant has been in operation, reliable average figures for recovery in the cyanide-plant are hard to obtain, for the only figures that are worth considering are those which are based on production of bullion during periods of at least six months.

Power.

The power for El Tigre is now generated at the power-plant of the Copper Queen Mining Co., in Douglas, Ariz., by two exhaust-steam turbo-generators. At the step-up transformer station in Douglas there are four 400-K. V. A. General Electric Co.'s transformers, any one of which may be held in reserve. The transmission-line, 65 miles long, is designed for 700 kw. capacity at 44,000 volts. The wires are No. 4 medium hard-drawn copper, and are spaced 6 ft. center to center in an equilateral triangle. The poles are of wood with treated butts, spaced for an average span of 200 ft. Each pole carries a small cross-arm below the transmission-wires for two No. 10 medium hard-drawn copper telephone wires. The longest span on the line is 1,650 feet.

At the step-down station four 320 K. V. A. transformers lower the tension to 440 volts for operating the various motors in the plant. The transmission-line and transformers were erected under the supervision of Sanderson & Porter, of New York. The efficiency of the transmission is at present about 91 per cent., owing to the amount of power that is used being considerably less than the full capacity of the line and transformers. The average cost of power is now only about one-third of the former cost when gas- and steam-engines at the mine were used.

Interruptions to the transmission from lightning occasionally happen in the summer season during the rains, but seldom exceed 1 hr. in duration.

Acknowledgment.

In conclusion, I wish to acknowledge the many courtesies and the hearty co-operation extended to me by the officers of the Lucky-Tiger Co. during the design and construction of its plant. In particular are my thanks due to James W. Malcolmson, consulting engineer, and to L. R. Budrow, manager, through whose kindness I am enabled to publish much of the operating data which appears in this paper, and from whom I received valuable criticism and advice during the work.

TABLE X.—*The Tigre Mining Co., S. A., Yzabal, Sonora, Mexico.*

ELECTRIC-POWER TESTS IN MILL AND CYANIDE-PLANT, AUGUST AND SEPTEMBER, 1911.

	Electric Horse-Power at Motor by Wattmeter.			
	Starting.	Normally Loaded.	Empty.	P. F.
				Per Cent.
Breaker-Plant.—Capacity 28 tons per hour (35-h-p. Allis-Chalmers motor, 850 r.p.m.).				
1 Style K, No. 4, Allis-Chalmers breaker set to 1.5 in.; 1 horizontal 32-in. sorting-belt 28 ft. c.c.; 1 24- by 27-in. revolving-drum ore-feeder; 1 shaking-grizzly 30 by 78 in.; 70 per cent. of ore dropping through the grizzly as fines	12.0	8.4	4.0	
No. 1 Concentrating-Mill.—Capacity 100 tons per day (150-h-p. Allis-Chalmers slip-ring type motor, 1,130 r.p.m., 14 line-shafts in mill, power-transmission from motor).				
Full load except Austin gyratory crusher, 5 vanners, and two Wilfley tables	215	144.7	.....	
4 4- by 5-ft. trommels, 1 pair 16- by 36-in. Cole-pattern roughing-rolls, 1 double-compartment bull-jig, 1 bucket-elevator 39 ft. c.c., 18-in. belt, 15-in. buckets	.....	32.2	.....	
1 pair 16- by 36-in. Cole-pattern roughing-rolls	.....	23.4	10.7	
4 double-compartment jigs 23.5 by 35.75 in.; also 1 three-compartment jig, 170 ⅝-in. strokes per minute	.....	16.0	.....	
1 pair 16- by 32-in. fine rolls	.....	18.0	8.0	
1 bucket-elevator 42 ft. c.c., 18-in. belt, 15-in. buckets, speed 360 ft. per minute	.....	4.0	.....	
2 4- by 5-ft. trommels ¼-in. round holes	.....	1.2	.....	
2 4- by 5-ft. trommels 1.5-mm. round holes	.....	0.9	.....	
1 5-ft. Huntington mill	.....	4.8	.....	
1 bucket-elevator 35 ft. c.c., 12-in. belt, 9-in. buckets, speed 360 ft. per minute	.....	3.2	.....	
1 9- by 10 in. Aldrich triplex return-water pump, 47 r.p.m	.....	7.5	.....	
10 No. 5 Wilfley tables and 1 shaking-launder	.....	10.8	.....	
1 10- by 54-in. Frenier sand-pump	.....	2.7	.....	
Machine-shop line-shafting with shaper and pipe-machine working	.....	2.6	.....	
11 6-ft. Frue vanners and 2 shaking-launders	.....	5.4	.....	
Line-shafting driving Wilfley tables and vanners	.....	.....	5.3	
Jack-shaft driving Huntington mills with coarse and fine rolls running empty	.....	.....	26.2	
Jack-shaft driving Huntington mills empty	.....	.....	3.2	
Line-shafting for crusher, jigs, elevators, and trommels, running empty	.....	.....	26.8	
Total line-shafting of mill running empty	.....	.....	48.3	
Ore-Sampling Plant.—Capacity 28 tons per hour, 1 10-h-p. Allis-Chalmers motor 850 r.p.m.				
1 revolving-cone sampler and ore-distributor, 1 Dodge sample-crusher 7 by 10 in., 1 Snyder sampler 27 in., 1 Sturtevant 2- by 6-in. roll-jaw sample-crusher, 1 30- by 78-in. shaking grizzly, ¾-in. spacing	.....	.....	3.48	85
Same running with load on inclined 14-in. conveying-belt assisting the motor	.....	3.2	.....	85
No. 2 Mill.—(1 75-h-p. Allis-Chalmers slip-ring induction-motor driving stamps and 1 50-h-p. synchronous motor driving tables, pump, and grinding-pan), capacity 100 tons.				
20 1,200-lb. stamps 102 7-in.-drops per minute	.....	44.4	.....	85
Line-shafting and 12-in. conveying-belt with tripper	.....	9.2	.....	85
Full load on motor	84	53.6	.....	85
Full load on synchronous motor with pump, tables, grinding-pan, and exciter	.....	21.4	.....	85
1 9- by 10-in. Aldrich return-water pump	.....	5.4	.....	
1 5-ft. grinding-pan for middlings from Wilfleys, 5 Wilfley No. 6 tables, 4 Deister slime-tables, 4 line-shafts, and exciter motor	.....	2.9	.....	
	.....	13.1	.....	

TABLE X.—*Continued.*

ELECTRIC-POWER TESTS IN MILL AND CYANIDE-PLANT.

	Electric Horse-Power at Motor by Wattmeter.			
	Starting.	Normally Loaded.	Empty.	P. F.
				Per Cent.
Cyanide-Plant.—Capacity 250 tons per day.				
A. Tailings Conveying- and Screening-Plant for Dumps-Tailings. Capacity 75 tons per day				
1 450-ft. 10-in. flexible tailings-conveyor, 1 3- by 5-ft. 6-in. disintegrating-trommel, 1 spiral conveyor, 1 bucket-elevator 25 ft. 5 in. c.c., 8-in. belt, 6-in. buckets, 1 22-in. by 5-ft. 6-in. mixing-trommel, with full load	10.7	5.0	.....	75
Shafting of above without load driven by 10-h-p. Allis-Chalmers motor	.....	.....	1.3	75
Conveying-belt only	.....	.....	2.8	
No. 2 flexible tailings-conveyor driven direct by 5-h-p. Allis-Chalmers motor	10.0	1.6	1.3	75
B. Tube-Mill Section (5 50-h-p. Westinghouse slip-ring motors direct connected to tube-mills, 1 35-h-p. Allis-Chalmers motor driving elevators, classifiers, and dewaterers).				
5 5- by 14-ft wet-grinding tube-mills, each	117	43.5	.....	75
5 42-in. drag-classifiers, 3 Dorr thickeners, 2 bucket-elevators 19 ft c.c., 18-in. belt, 15-in. buckets, speed 400 ft. per min., 2 bucket-elevators same as previous but 40 ft. c.c., 4 line-shafts	16	10.7	.....	75
4 elevators plus line-shaft friction	.....	9.6	.....	
5 classifiers with 3 Dorr thickeners and three line-shafts	.....	1.1	.....	
C. Upper Pump-House (1 25-h-p. Allis-Chalmers motor and 1 line-shaft).				
1 6- by7-in. return-solution Aldrich triplex pump and 1 7- by 9-in. Aldrich triplex return-water pump plus line-shaft friction	.....	14	.....	75
Motor and shafting	3.2	.....	2	
D. Treatment Section.				
1 Sullivan WI 16- by 14-in. duplex belt-driven air-compressor, driven by 100-h-p. Allis-Chalmers motor, compressor making 166 r.p.m. at 30 lb. pressure	.....	90	.....	
1 30- by 12-ft. mechanically-agitated slime-storage-tank and two Tigre slime-thickening tanks with two line-shafts	18.7	6.9	.....	75
30- by 12-ft. agitation-tank only with one line-shaft	.....	5.9	.....	
2 24- by 12-ft. Tigre thickeners and two line-shafts	.....	1.0	.....	
Line-shafting only	.....	.....	0.3	
E. Filter-Plant and Precipitation-Pumps.				
1 class "NE-1" Ingersoll-Rand straight-line belt-driven air-compressor, size 8 by 8 in., 210 r.p.m. at 60 lb. average pressure, furnishing 96 cu. ft. free air per min. to Kelly filter-plant, driven by line-shaft from 50-h-p. Allis-Chalmers motor	107.2	15.5	7.5	75
2 6- by 7-in. Aldrich triplex precipitation-pumps.	.....	11.2	.....	
Spiral Ribbon conveyor for discharge of residues from Kelly filters, not in continuous use	34	4	0.7	75
1 20- by 10-ft. mechanically-agitated excess slime-storage tank for filling Kelly filters, with one small pyramid return-water pump for counterweight-tanks	16	2.7	0.5	75
F. Pumps, etc., in occasional use only.				
1 2-in. centrifugal return-water pump	43	20.7	10.7	85
1 1.5-in. centrifugal sand-pump for elevating thickened slime from redwood tanks	20	3.5	0.3	
Miscellaneous Power.				
1 3-h-p. motor for assay-office	.....	2	.....	
1 3-h-p. motor driving ice-plant, capacity 1,000 lb. of ice per day	10.7	3.6	1.6	
1 15-h-p. back-geared Westinghouse motor for incline hoist, capacity 3 tons at 200 ft. per minute	.....	18	.....	85
1 10-h-p. Allis-Chalmers motor for carpenter-shop	37	6.7	3.3	
1 14- by 9- by 16-in. WH 2 Sullivan belt-driven air-compressor at 80 lb. pressure making 140 r.p.m.	.....	65.5		
at 85 lb. pressure making 140 r.p.m	.....	67.5		
at 90 lb. pressure making 140 r.p.m	.....	69.8		
at 95 lb. pressure making 139 r.p.m	.....	71.8		
Line-shaft with clutch to mine air-compressor thrown out	.....	.....	4.1	

TABLE X.—*Continued.*

ANALYSIS OF POWER-CONSUMPTION DURING AUGUST, 1911.

	Total Monthly El. H-P. Hours.	Average Power when Running.	Total Average Power.
		El. H-P.	El. H-P.
Sorting and Ore-Conveying, including lights	1,283.4	3.45	3.45
No. 1 Mill.—Stage-Crushing			167.10
Crushing, including lights	42,360.0	66.50	
Regrinding, including lights	5,431.0	8.50	
Concentrating, including lights	55,437.9	90.10	
No. 2 Mill.—Gravity Stamps for Crushing			86.20
Crushing, including lights	37,667.2	62.20	
Regrinding, including lights	5,431.0	3.60	
Concentrating, including lights	9,147.7	20.40	
Cyanide-Plant			401.60
Conveying and screening tailings	2,418.0	7.60	
Tube-milling, including lights	136,743.6	198.30	
Classification, elevation, and dewatering	22,512.6	52.10	
Treatment, including lights	72,108.6	98.70	
Filtration, including lights	17,912.0	25.20	
Precipitation, including lights	8,861.0	13.70	
Melting, lights, and electric drying-oven	3,576.0	6.00	

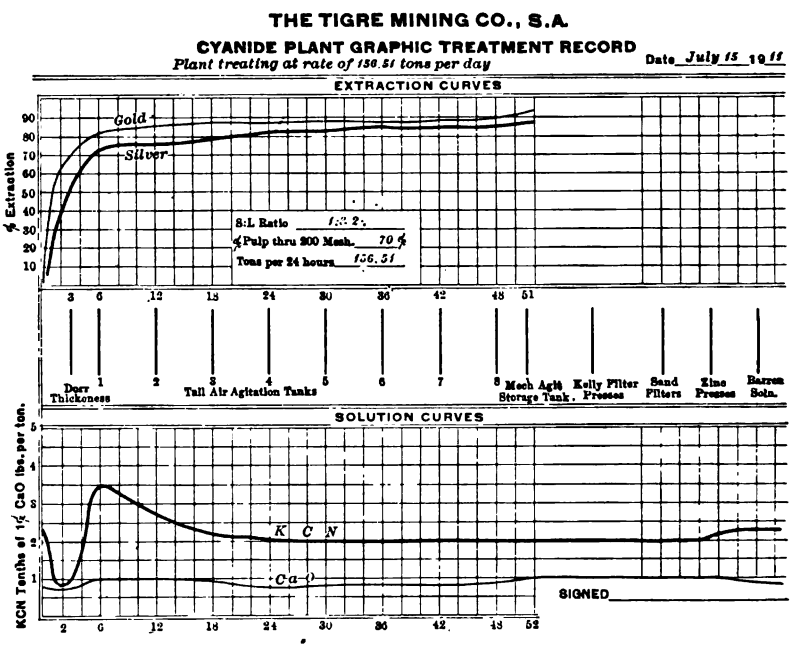


FIG. 10.—RECORD OF CYANIDE-TREATMENT.
(Size of original form, 8.5 by 10.5 in.).

The test report and the diagram, Fig. 10, are copied from the forms used during the starting of the plant. The standards given on the test report were predetermined figures based on the results of experimental work, and calculations used in making up the flow-sheet for the cyanide-treatment.

Form 92.

[Size of original form, 8.5 by 10.5 in.]

THE TIGRE MINING CO., S. A.

CYANIDE PLANT TEST REPORT.

Printed figures [here shown in heavy-faced type] denote standards for normal operation on 250 Ton basis.

Date *July 15, 1911.*

156.51

Ton–2,000 lbs.

		Overflow Concentrates Bins.	Disch. Settlings From Pond.	Classifiers. No's. 1 & 2. Feed.	Classifiers. No's. 1 & 2. Sand.	Classifiers. No's. 1 & 2. Slime.	Classifiers. No's. 3, 4 & 5. Feed.	Classifiers. No's. 3, 4 & 5. Sand.	Classifiers. No's. 3, 4 & 5. Slime.	Tube Mills, No's. 1 & 2.	Tube Mills, No's. 3, 4 & 5.	Dorr. Thickeners, No's. 1 & 2.	Dorr. Thickener, No. 3.	Wilfley Filter Cake.	Pulp Elevators, No's. 3 & 4.	No. 1 Agitation Tank.	No. 2 Agitation Tank.	No. 3 Agitation Tank.	No. 4 Agitation Tank.	No. 5 Agitation Tank.	No. 6 Agitation Tank.	No. 7 Agitation Tank.	No. 8 Agitation Tank.	Tigre Thickeners, No's. 1 & 2.	Agitation Storage Tank.	Discharged Filter Cakes.	Strong Stock Solutions.	Weak Wash Solution.	Strong Pregnant Solution.	Weak Pregnant Solution.
												PULP.															SOLUTIONS.			
Siz. Tests.	+ 100 Mesh.			*56.0*	*91.1*					*50.4*	*51.7*																			
	+ 150 Mesh.			*3.2*	*3.7*					*10.4*	*18.3*																			
	+ 200 Mesh.			*2.1*	*2.2*	*9.5*			*36.0*	*4.5*	*11.3*				*39.1*	*28.2*	*26.8*	*28.2*	*31.8*	*30.2*	*28.2*	*29.4*	*29.7*							
	— 200 Mesh.					*90.5*			*64.0*	*34.7*	*18.0*				*60.9*	*71.8*	*73.2*	*71.8*	*68.2*	*69.8*	*71.8*	*70.6*	*70.3*							
				38.7	*3.0*	**90.**			**90.**			**90.**	**90.**		**90.**															
	Sp. Gr.			*1.105*	*1.50*	*1.04*		*1.45*	*1.14*	*1.50*	*1.45*	*1.45*	*1.22*	×	*1.24*	*1.16*	*1.10*	*1.18*	*1.18*	*1.20*	*1.20*	*1.20*	*1.20*	No.1 & 2 1=1.38 2=1.20	*1.31*					
					1.48	**1.04**		**1.48**	**1.11**			**1.43**	**1.21**		**1.25**										**1.32**					
	S. to L. Ratio.			*1:6*	*1:1–*	*1:17*		*1:1+*	*1:5–*			*1:1–*	*1:2.3*		*1:2.2*	*1:4*	*1:3.5*	*1:3.5*	*1:3.5*	*1:3.2*	*1:3.2*	*1:3.2*	*1:3.2*		*1:1.6*					
				1:6		**1:17**			**1:5**			**1:1**	**1:2.4**		**1:2**										**1:1.5**					
	Per Cent. Moisture.				**50.**			**50.**				**50.**		**30.**												**40.**				
	Per Cent. K. CN.														*.080*	*.35*	*.275*	*.210*	*.190*	*.200*	*.205*	*.205*	*.210*		*.210*		*0.23*	*0.16*	*0.20*	*0.16*
																0.35											**0.26**	**0.05**	**0.24**	**0.05**
	Ca. O. pounds.														*.90*	*1.05*	*1.00*	*.95*	*.75*	*.80*	*.80*	*.80*	*.85*		*1.20*				*1.00*	*.85*
																2.4											**3.**			
Assays.	Gold. oz.	*Average of all Tailings for day.*		*.079*							*.03*	*.05*	*.018*			*.015*	*.01*	*.01*	*.01*		*.01*	*.008*	*.008*		*.006*	*.005*	*.004*	*.001*	*.024*	*.014*
																									.005		**.005**	tr.		
	Silver. oz.			*17.22*							*10.47*	*20.95*	*5.58*			*4.69*	*4.09*	*3.59*	*2.99*		*2.69*	*2.69*	*2.69*		*2.39*	*2.47*	*0.42*	*0.21*	*5.15*	*3.05*
																									.70		**.30**	**.08**		
	Values. U. S. Cy.			*$10.71*																					*$1.39*	*$1.41*	*$0.24*	*$0.13*	*$3.06*	*$1.81*
																									.45		**.25**	**.04**		
Pr. Ct. Ex.	Gold.										*62.*					*81.2*	*87.5*	*89.5*	*89.5*		*89.5*	*89.8*	*89.8*		*92.4*					
																.57	**.78**	**87.**	**91.**	**92.**	**92.5**	**93.5**	**94.**							
	Silver.										*39.2*					*72.7*	*76.3*	*79.3*	*82.5*		*84.4*	*84.4*	*84.4*		*86.2*					
																41.	**65.**	**77.**	**83.5**	**86.**	**87.5**	**89.**	**90.**							
	Values. U. S. Cy.															**43.5**	**67.**	**79.**	**85.**	**87.5**	**89.**	**90.**	**90.5**		*87.0* Pr.Ct.	*86.8* Pr Ct.				
Av. RateFl.	Gals. per min.			**192**								**19**	**110**		**125**												**83**		**83**	**125**
	Total Tons per 24 hrs.			**1148**								**112**	**659**		**750**												*338*		*338*	*660*
																											500		**500**	**750**
	Dry Tons per 24 hrs.			**164**			**340**			**164**	**340**	**56**	**194**	**56**	**250**															

Total Tailings Treated for day = 156.51

Signed _____

Cyanide Foreman.

Electrical Fume-Precipitation.

BY F. G. COTTRELL, SAN FRANCISCO, CAL.

(New York Meeting, February, 1912.)

ABOUT a year and a half ago, at the San Francisco meeting of the American Chemical Society, in connection with the excursions to local smelting-works, I had occasion to show some lantern-slides illustrating the recent commercial development of electrical methods for precipitating the suspended matter from smelter-smoke, the underlying phenomena of which were clearly brought to the attention of the technical public by Sir Oliver Lodge as long ago as 1884. In the *Journal of Industrial and Engineering Chemistry*, August, 1911, most of these views were reproduced, together with an account of the work from its beginning up to July, 1911.

To-day I wish particularly to show you some of the further developments of the work since that time; but in order to make these clear they will be prefaced by a brief abstract of the article above referred to.

The removal of suspended particles, from gases, by the aid of electric discharges is by no means a new idea. As early as 1824 we find it suggested by Hohlfeld [1] as a means of suppressing ordinary smoke, and again a quarter of a century later by Guitard.[2] These suggestions, which do not seem to have stimulated any practical study of the question, were soon entirely forgotten, and only brought to light again by Sir Oliver Lodge,[3] many years after he himself had independently rediscovered the same phenomena and brought them to public attention [4] in a lecture before the Liverpool Section of the So-

[1] "Das Niederschlagen des Rauchs durch Electricität Hohlfeld," *Kastner Archiv Naturlehre*, vol. ii., pp. 205 to 206 (1824).

[2] C. F. Guitard, *Mechanics' Magazine* (Nov., 1850).

[3] Historical Note on Dust Electrification and Heat, O. J. Lodge, *Nature*, vol. lxxi., p. 582 (1905).

[4] The Electrical Deposition of Dust and Smoke, with Special Reference to the Collection of Metallic Fume, and to a Possible Purification of the Atmosphere, *Journal of the Society of Chemical Industry*, vol. v., No. 11, pp. 572 to 576, with appended bibliography (Nov. 29, 1886).

ciety of Chemical Industry, Nov. 3, 1886. The first recorded attempt to apply these principles commercially appears to have been made at the Dee Bank Lead Works. The general principle of electrical precipitation of suspended matter was at this time patented by Alfred O. Walker, of the above firm, in several countries,[5] but these patents have long since expired. The apparatus was installed in 1885 by the works-manager, W. M. Hutchings, with the co-operation of Professor Lodge, and briefly described by the former,[6] just before its completion, as consisting of a system of metallic points situated in the flue from the lead-furnaces, and excited from two Wimshurst influence-machines, with glass plates 5 ft. in diameter, each machine being driven by a 1-h-p. steam-engine.

The apparatus undoubtedly did not in practice fulfill expectations, as we find nothing further of it in the literature. The most apparent weakness of the project lay, perhaps, in the reliance on the Wimshurst machine, which had then just been brought out, and from which a great deal more was anticipated[7] than has been justified by experience, at least as far as commercial applications are concerned.

None of these suggestions and patents had been successful in commercial operation when, about 1906, occasion was taken to repeat Lodge's early experiments, and to endeavor to reduce to engineering practice, under more favorable conditions, the principles and processes developed by Lodge and Walker.

The precipitation of suspended matter in gases or liquids may be accelerated by either the alternating or the direct electric current. The former acts chiefly by agglomerating the suspended particles so that, constituting larger bodies, they settle by gravity more rapidly. Such an application has been made in Europe, it is claimed, by sending powerful Hertzian waves into foggy air. Another is the use of the alternating

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[5] Great Britain, 11,120, Aug. 9, 1884; Belgium, 68,927, May 19, 1885; Spain, 7,211, July 10, 1885; Germany, 32,861, Feb. 27, 1885; Italy, 18,007, Mar. 31, 1885; United States, 342,548, May 25, 1886.

[6] *Berg- und Hüttenmännische Zeitung*, vol. xliv., No. 25, pp. 253 to 254 (June 19, 1885).

[7] A. O. Walker, *Engineering* (London), vol. xxxix., pp. 627 to 628 (June 5, 1885). G. Tissandier, *Electrician* (London), vol. xvii., p. 33 (May 21, 1886).

current devised by Buchner Speed and me for separating emulsified water from crude California petroleum.[8]

But for the voluminous gases of smelters, moving rapidly through flues—or even expanded into dust-chambers—this agglomerating and settling process is too slow; and recourse must be had to the direct current. If a needle-point, connected to one side of a high-potential direct-current line, be brought opposite to a plate connected to the other side of the line, the space between, and any insulated body contained in it, become highly charged with electricity of the same sign as the needle, whether positive or negative; and such a body, if free to move, will be attracted to the plate of opposite sign. Suspended particles of fume may thus be precipitated, not slowly by gravity, but instantly by electricity, upon electrodes.

To make this action commercially practicable, the first step was to get an economic and effective source of high-tension direct current. In the installations here described, this was done by transforming an ordinary alternating current up to 20,000 or 30,000 volts, and then commutating to an intermittent direct current, by means of a rotary contact-maker driven by a synchronous motor. This high-tension direct current is applied to a system of electrodes in the flue carrying the gases to be treated.

It has been largely in the development of practical forms and arrangements of electrodes to meet the exacting and varied conditions of large-scale operations in different installations that the most difficult as well as the most interesting problems have presented themselves and have gradually been worked out to successful solutions.

The first experiment on a working-scale was made at the Hercules powder-works at Pinole, Cal. The gases treated contained about 4 per cent. by volume of dry, gaseous sulphur trioxide, and were brought into contact with water, which, combining with the sulphur trioxide, formed the far less volatile sulphuric acid, which immediately separated as a dense white cloud of suspended particles, so fine as to constitute one of the

[8] Dehydration of Crude Petroleum, a New Electrical Process, by Arthur T. Beazley, *The Oil Age*, vol. iii., No. 13, p. 2 (April 21, 1911). Water Emulsions in Crude Petroleum, by Allen C. Wright, *Western Engineering*, vol. i., No. 3, pp. 207 to 210 (June, 1912).

most difficult of all materials to remove by filtration, but easily and successfully precipitated by the electric current.

The next installation was made at the Vallejo Junction works of the Selby Smelting & Lead Co., where the stack of the refinery discharged the mists which escaped from the boiling sulphuric acid used to dissolve silver. These were successfully treated, and the electrical precipitation process has been in steady commercial operation there ever since.

The problem next attacked was the treatment of the gases from the pyrites- and matte-roasters of the same plant. Here the material to be removed was a mixture of solid dust and fume, with liquid sulphuric acid. The latter often exceeded in weight all the solids in the gas. Moreover, the gases were highly charged with water-vapor, which diluted the acid and produced a muddy deposit. These features required a new construction, by which the gases were treated in flues of sheet-lead.

A later and larger installation was made at the Balaklala smelter at Coram, Shasta county, Cal., where more than 250,000 cu. ft. of gas per minute was treated.

The work at Balaklala may fairly be likened to one of those "successful" surgical operations where the patient incidentally dies. The process worked, and technical difficulties were one after another overcome and efficiency steadily increased during the months of operation up to the very end, but general conditions did not justify the continued operation of the smelter, and it finally closed.

The installation served, however, to demonstrate thoroughly that there was nothing fundamentally impracticable about applying the method to plants of any desired size. In the paper in the *Journal of Industrial and Engineering Chemistry*, of which the foregoing is a brief and partial abstract, details (including drawings and photograph) of the various plants may be found.

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After the Balaklala, the next large problem to be actively undertaken was the equipment of the plant of the Riverside Portland Cement Co., in southern California. The orange-growers of the surrounding district had complained of the dust thrown out from the rotary kilns, and this company was very desirous of abating the nuisance.

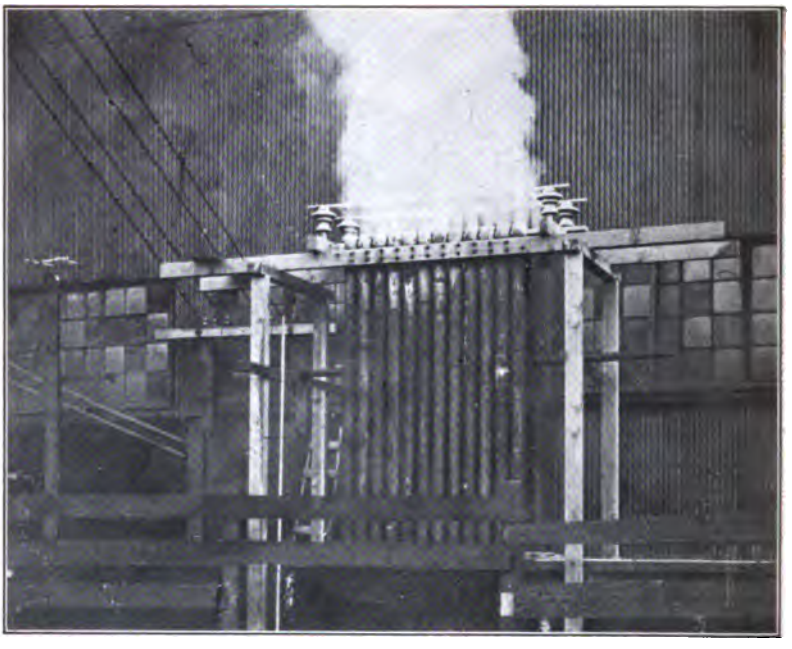

FIG. 4.—EXPERIMENTAL TUBULAR TREATER AT THE GARFIELD PLANT BEFORE APPLICATION OF CURRENT.

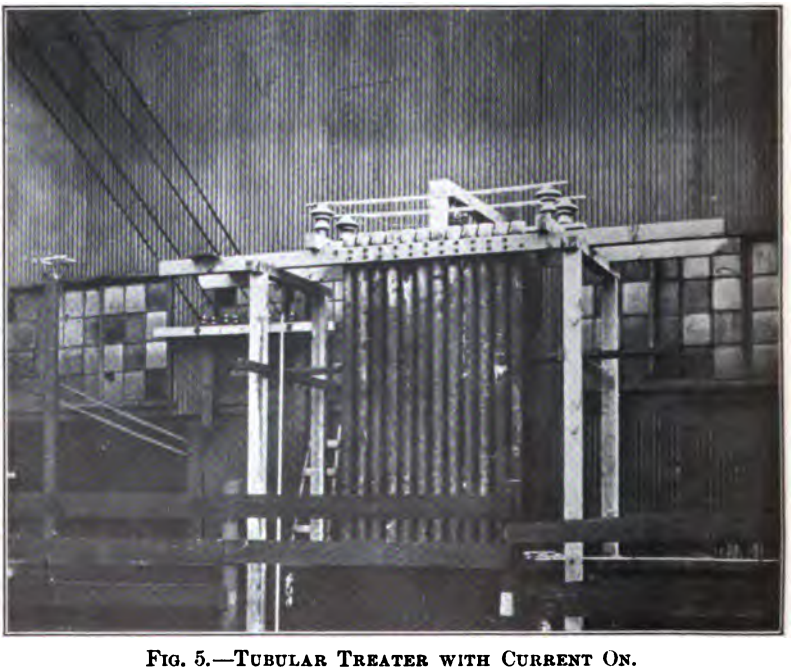

FIG. 5.—TUBULAR TREATER WITH CURRENT ON.

of the patent-rights covering the process in this country are being devoted.

As mentioned in the introduction, this work grew from experiments started in the Chemistry Department of the University of California. When it became necessary to extend these to operations on a commercial scale, it was three alumni of that department, E. S. Heller, Dr. Harry East Miller, and Prof. Edmond O'Neill, who furnished the sinews of war and helped to guide the business administration, under an informal mutual understanding that, as soon as the venture repaid the investment with reasonable interest, a considerable portion at least of the remaining patent-rights should be turned over either to the University or to some other academic institution, as the nucleus of an endowment for scientific research.

By the time the business had reached this degree of financial success, it was no longer local in its application; and it was felt that it should be in the hands of a more widely representative institution than a State university. Consultation with the officers of several of our national scientific and engineering societies made it clear that none of these societies was organized to administer adequately such a business; but finally, through Director Holmes, of the Bureau of Mines, the matter was brought to the attention of the Smithsonian Institution, and a formal offer of the patent-rights still remaining unsold was made to that body.

In consultation with the Board of Regents of the Institution it was finally decided most expedient to create a separate board of administration to manage business matters in connection with licensing and further development of the patents, and turn over the net profits to the Smithsonian or other scientific and educational institutions or societies for use in scientific research. This Board has recently been organized under the name of the "Research Corporation," with offices at 63 Wall Street, New York City, and will be conducted like any other business corporation, save that no dividends are paid to stockholders, the whole of the net profits being devoted to scientific research.

Already a number of other patents are being offered to the corporation, both from this country and abroad, for the move-

ment has no geographical limits, either as regards sources of revenue or expenditure of income.

As the corporation will soon issue detailed literature descriptive of its aims, objects, and methods, it will be needless to detain you here with these, but your interest and encouragement are earnestly bespoken in behalf of this new project.

An Early Discovery of Fullers' Earth in Arkansas.

BY J. C. BRANNER, STANFORD UNIVERSITY, CAL.

(New York Meeting, February, 1912.)

DURING the past two or three years I have seen statements regarding the first discovery of fullers' earth in this country that seem to require correction or modification. One appears again in the *Mineral Resources of the United States for* 1909, part II., p. 738, where it is said that "fullers' earth was first discovered in the United States at Quincy, Fla., in 1893."

In this connection the following bit of unpublished history concerning fullers' earth in the State of Arkansas is worthy of attention.

In April, 1891, when I was State Geologist of Arkansas, a Swede named John Olsen, formerly a copper-miner in Michigan, but then living south of Little Rock, brought me a sample of what he thought was fullers' earth. He reported that it was from near the town of Alexander, the locality being 1 S. 13 W., section 8, SW. quarter of the SE. quarter. My chemist, Dr. R. N. Brackett, analyzed the material, and reported:

		Per Cent.
Silica,	SiO_2	63.19
Alumina,	Al_2O_3	18.76
Iron,	Fe_2O_3	7.05
Lime,	CaO	0.78
Magnesia,	MgO	1.68
Potash,	K_2O	0.21
Soda,	Na_2O	1.50
Loss on ignition,		7.57
		100.74

Water at 110° to 115° C. = 7.84.

Another sample from about the same locality, also brought by John Olsen, on analysis by Dr. Brackett gave:

		Per Cent.
Silica,	SiO_2	62.92
Alumina,	Al_2O_3	18.24
Iron,	Fe_2O_3	7.62
Lime,	CaO	0.76
Magnesia,	MgO	1.77
Potash,	K_2O	0.54
Soda,	Na_2O	1.44
Loss on ignition,		7.55
		100.84

Water at 110° to 115° C. = 6.95.

Analyses were also made of samples of imported fullers' earth, and the similarity between the domestic and the foreign materials was so close that it was thought worth while to have practical experiments made with the Arkansas earth.

On May 12, 1891, the results of these analyses were reported to A. M. Littlejohn, of Little Rock, Ark., who was at that time, as manager or otherwise, connected with the Southern Cotton Oil Co., which was using these clays for refining cotton-seed oil in its mills at Little Rock.

On Sept. 30, 1891, I went personally to Alexander and to Niemeyer and visited the region from which these earths had been reported. I collected a sample on 1 S. 13 W., section 8, SW. quarter of the SE. quarter, which, analyzed by Dr. Brackett, gave:

		Per Cent.
Silica,	SiO_2	64.38
Alumina,	Al_2O_3	17.29
Iron,	Fe_2O_3	8.25
Lime,	CaO	1.11
Magnesia,	MgO	0.80
Potash,	K_2O	1.41
Soda,	Na_2O	0.42
Loss on ignition,		6.95
		100.61

Water at 110° to 115° C. = 9.02.

It will be noted that these analyses are as nearly alike as could be expected. Of course, they were not, and could not be, regarded as settling the question whether the earths were or were not fullers' earths. But they were considered as quite favorable to such a theory, and Mr. Littlejohn was encouraged to give the earth in question a thorough practical test. For some time the Arkansas material was used in the oil-works.

Just how long it was used, I do not know, for Mr. Littlejohn, who had charge of the works, died in December, 1891, and I am unable to learn any of the details of his experience. Mr. Olsen, however, reports that the Southern Cotton Oil Co. finally gave up its use.

I recall the fact that it seemed to me, at the time when the experiments were being made, that the methods used in obtaining the earth must of a necessity lead to failure. This fullers' earth is the weathered portions of nearly horizontal beds of Tertiary clays that run through the low hills over a large area in the region S. and SW. of Little Rock. According to the miners' preconceived notion that ore must "improve with depth," it was expected that after the weathered edges of the clays had been passed the fullers' earth must likewise improve. I remember also that, just when Mr. Littlejohn thought the earth was going to be a great success, the miners began to congratulate themselves that the stuff was looking "good enough to eat," and consequently that it must be improving. Almost immediately, Mr. Littlejohn reported that it was not working well. To me the reason seemed very clear; the quarrymen had left the weathered zone behind, and had entered the unaltered clays.

I understand that the fullers' earth business in Arkansas is now well established and constantly growing. The only object of the present brief statement is to call attention to the fact that fullers' earth was known and was already being mined in the State of Arkansas as early as 1891.

Even since the present paper was written, an advance chapter of the *Mineral Resources for* 1910, upon the production of fullers' earth, repeats, at p. 4, the statement that the first "discovery of fullers' earth in this country" was at Quincy, Fla., in 1893.

The Decomposition of Metallic Sulphates at Elevated Temperatures in a Current of Dry Air.

BY H. O. HOFMAN AND W. WANJUKOW, MASS. INSTITUTE OF TECHNOLOGY, BOSTON, MASS.

(New York Meeting, February, 1912.)

I. Introduction.

In the metallurgical treatment of most metallic sulphides it is usually necessary to carry on a roasting-operation. In some cases the raw ore will have to be roasted; in others the intermediary product, matte, produced in smelting the ore; again, both ore and matte may have to be subjected to oxidation at elevated temperatures.

Most metallic sulphides, when given an oxidizing-roast, pass through the state of sulphate before they are converted into oxides. All metallic sulphates are completely decomposed by heat, with the exception of those of lead and bismuth, but the temperatures at which decompositions take place vary with the metals. Some metallic sulphates are converted directly into oxides, others first form basic salts. Alkali-earth sulphates show a behavior similar to that of metallic sulphates, while alkali sulphates do not part with their sulphur trioxide even at elevated roasting-temperatures.

The older data on the decomposition of metallic sulphates by heat are only approximations. Thus Kerl[1] gives the order in which metallic sulphates are decomposed with increasing temperatures as (Ag_2, Fe, Cu, Zn, Ni, Co, Mn, Pb) SO_4. More recent statements give definite temperatures and modify the formerly accepted order of decomposition. But the temperatures given by various observers do not agree. The reasons for this are to be found mainly in the method and manner of carrying out experiments.

Four methods have been used: 1. Heating in an evacuated tube; 2, heating in a tube closed at one end, without air-cur-

[1] *Grundriss der Metallhüttenkunde*, p. 70 (Leipzig, 1881).

rent; 3, heating in a crucible with air-current; and 4, heating in an open tube with air-current.

1. *Heating in an Evacuated Tube.*—This method is the one followed by Wöhler-Plüddemann-Wöhler.[2] A tube charged with the sulphate to be examined is evacuated, heated to different temperatures, and held at each temperature until equilibrium has been reached in the reversible equation $MSO_4 \rightleftharpoons MO + SO_3$. The pressure exerted by the SO_3, resp. $SO_2 + O$, set free at each temperature and read in millimeters of quicksilver on a gauge, furnishes the means for ascertaining the degree of decomposition of the sulphate. The decomposition at atmospheric pressure is the one at which the pressure of the gas in the tube has reached 760 mm. In order to determine whether the sulphate by heating is converted direct into oxide or first forms one or more basic salts, it is necessary to heat the normal sulphate first to a given temperature in a current of air to expel some of the SO_3, and then in vacuo for determining the dissociation-pressures at the same temperature to which the normal sulphate had been brought. If the pressure-readings of both experiments give the same figures, no basic salt has been formed up to the temperature to which the normal salt had been heated in a current of air. In this manner, samples of normal salt will have to be heated in air to different temperatures, and the partial decompositions finished in vacuo. A comparison of the dissociation-curve of the normal salt with those of the partly decomposed salts will show at a glance whether basic salts are formed or not. They investigated the following metallic sulphates: $Fe_2O_3.3SO_3$; $Al_2O_3.3SO_3$; $Cr_2O_3.3SO_3$; $CuO.SO_3$; $Ce_2O_3.3SO_3$; $ThO_2.2SO_3$; $TiO_2.SO_3$; $ZnO.SO_3$. The data for the sulphates of metallurgical interest have been plotted in Fig. 1, in which the ordinate gives the degrees centigrade scale, and the abscissa the pressures of SO_3 in millimeters of quicksilver. In order to translate the pressures of SO_3 into per cent. by volume of SO_3, and thus render the data comparable with roasting-gases, a second scale has been drawn on the abscissa, in which 760 mm. pressure has been made equal to 100 divisions, presupposing that SO_3 is not split into SO_2 and O; thus 1 volume of SO_3

[2] *Berichte der deutschen chemischen Gesellschaft*, vol. xli., pt. 1, No. 4, p. 703 (1908).

will equal 7.6 mm. pressure SO_3. These data, while of much scientific interest, cannot be well applied to the decomposition of metallic sulphates in a roasting-furnace, as in the latter the gases are drawn off continuously, and the partial pressure of SO_3 is so small as to have practically no influence whatever upon the decomposition.

2. *Heating in a Tube Closed at One End without a Current of Air.*—This method has been recently used by Friedrich and Blickle.[3] The apparatus consists of a vertical porcelain test-tube,

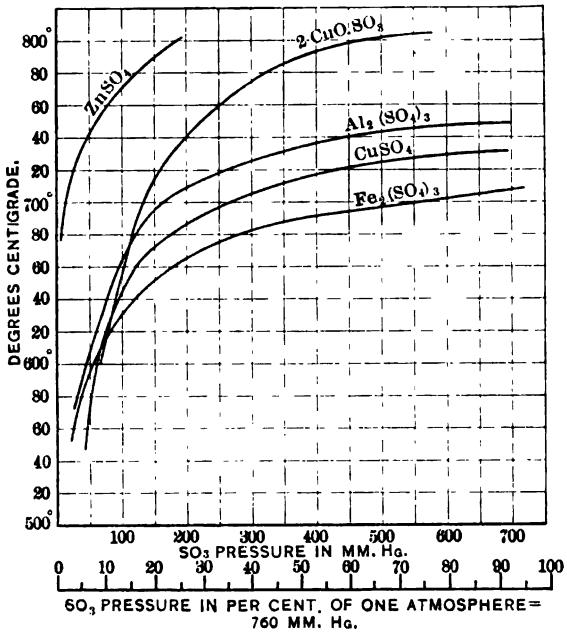

FIG. 1.—DECOMPOSITION OF SOME METALLIC SULPHATES IN AN EVACUATED TUBE.

3.937 in. long, 0.7078 in. outer and 0.5905 in. inner diameter, heated in a suitable furnace. Into the tube is lowered by means of platinum wires a platinum cup, 0.5905 in. high, 0.3543 in. diameter, with walls 0.006 in. thick, to receive from 0.5 to 1.2 g. of metallic sulphate to be decomposed. A tube holding the thermo-couple is introduced into the substance. This is heated from room-temperature to 1,100° C. in about

[3] *Metallurgie*, vol. vii., No. 11, p. 323 (June 8, 1910); *Stahl und Eisen*, vol. xxxi., No. 47, p. 1909 (Nov. 23, 1911).

1 hr.; the retardations due to absorption of heat by decomposition are noted. The results obtained are given in Table I.

TABLE I.—*Decomposition of Some Metallic Sulphates in a Tube Closed at One End.*

Metallic Sulphates.	Temperatures of Decomposition.	Products of Decomposition.	Transformations of Original Sulphates.	Fusions of Original Sulphates.
	Degrees C.		Degrees C.	Degrees C.
$Fe_2(SO_4)_3$	705	Fe_2O_3		
$CuSO_4$	740	$2CuO.SO_3$		
$Al_2(SO_4)_3$	770	Al_2O_3		
$NiSO_4$	840	NiO		
$ZnSO_4$	About 840	$3ZnO.2SO_3$	740	
$2CuO.SO_3$	845	CuO		
$CoSO_4$	880	CoO		
$3ZnO.2SO_3$	935	ZnO		
$MnSO_4$	1,030	?	860	700
CuO	About 1,040	$(CuO.Cu_2O)$		
Ag_2SO_4	1,085	Ag	410	660
$PbSO_4$	.....	.....	850	

A comparison of the data with those obtained by other investigators will show that they are from 140° to 240° C. too high. The two leading reasons for this discrepancy are, that the decomposition takes place in an atmosphere of SO_3, respectively SO_2 and O, the partial pressures of which call for higher temperatures of dissociation than when heating in an air-current, as here the partial pressures of the products of decomposition are a negligible quantity; and, secondly, that the short time of 1 hr. given to each of the experiments must allow the temperature to rise above the critical point, considering the low conductivity for heat[4] of the sulphates. The temperatures obtained ought to correspond to those observed with an evacuated tube in which the pressure of SO_3, respectively $SO_2 + O$, has reached 760 mm.; and this is approximately the case.

3. *Heating in a Crucible with Air-Current.*—The method has been followed by Bradford.[5] He obtained satisfactory results with the sulphates of iron, copper, and silver, by using a small quantity of substance and by placing the thermo-couple, in contact with substance, near the wall of the crucible, as it is here that the dissociation begins. If much substance is used, or if the

[4] H. Lescœur, *Comptes rendus de l'Académie des Sciences*, vol. cii., No. 25, p. 1466 (1886).

[5] *Trans.*, xxxiii., 58 (1902).

couple is placed in the center of the crucible, the temperature read for the beginning of a dissociation will be too high, on account of the low conductivity for heat of the substance.

4. *Heating in an Open Tube in a Current of Air.*—This method is the one which resembles more closely than any other the roasting-operation on a large scale. The substance to be tested is placed in a boat inserted into a horizontal tube impervious to gases; the thermo-couple is held near the boat or the substance; the tube is heated gradually, usually by electrical means; a slow currrent of purified and dried air is blown through and carries away the gaseous products of decomposition; the latter can be tested chemically whenever this seems desirable.

There are two ways of determining the beginning of a retardation caused by dissociation.

One is to heat to a given temperature, holding the thermo-couple near the boat; to keep the temperature unchanged until any loss in weight of the substance has become constant, and to watch at the same time any change in the indicator, held in a railroad-tube, through which the gases have to pass. If a decomposition has taken place, another sample of the substance is treated in the same manner as the first, but to a lower temperature. In this way the range of temperature is narrowed down until the actual dissociation-temperature has been accurately ascertained. The loss in weight will show to what extent the substance has been decomposed; a chemical analysis may be necessary, if a chemical change other than expulsion of gaseous component has taken place. This method has been followed by Hofman with zinc sulphate,[6] Hofman-Mostowitsch with calcium sulphate,[7] Mostowitsch with barium sulphate,[8] Warlimont with iron, copper, and nickel sulphates,[9] Kothny with pyrite,[10] Landis with iron, copper, and zinc sulphates,[11] Mostowitsch with zinc sulphate,[12] and Barth with cobalt sul-

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[6] *Trans.*, xxxv., 811 (1904).

[7] *Trans.*, xxxix., p. 628 (1908); xl., p. 807 (1909).

[8] *Metallurgie*, vol. vi., No. 14, p. 450 (July 22, 1909).

[9] *Metallurgie*, vol. vi., No. 3, p. 83 (Feb. 8, 1909).

[10] *Oesterreichisches Jahrbuch*, vol. lviii., pp. 97, 350 (1910); *Metallurgie*, vol. viii., No. 13, p. 389 (July 8, 1911).

[11] *Metallurgical and Chemical Engineering*, vol. viii., No. 1, p. 22 (Jan., 1910).

[12] *Metallurgie*, vol. viii., No. 24, p. 763 (Dec. 22, 1911).

phate.[13] It has given accurate results as long as the necessary time has been given to the work.

The other method is to make first a blank test, using the amperage necessary to reach a desired temperature, with the thermo-couple held in the boat, but out of contact with the substance; to record the rise of temperature for given intervals of time, 30 or 60 sec., and to plot the data which will furnish a smooth curve. The next step is to charge the boat, raise the temperature in the same manner as with the empty boat, record the rise in temperature for the same intervals of time, and to plot the data. Any deviation of the second curve from the first will show that some change has taken place; there will have been a transformation if the weight of the substance has remained constant, a chemical change if the weight has increased or decreased. The method was devised by Wanjukow[14]; the abstract[15] contains some errors in his investigations upon cupric sulphate (see p. 548). As it is the method followed in the present research, the following example is given with some detail to make clear the mode of operation.

TABLE II.—*Dehydration of* $FeSO_4 + 7H_2O$.

CURRENT: 7 AMPERES AT 110 VOLTS.

Time-Interval.	Galvanometer Deflection.		Time-Interval.	Galvanometer Deflection.		Time-Interval.	Galvanometer Deflection.	
	Blank Test.	Test With $FeSO_4 + 7H_2O$.		Blank Test.	Test With $FeSO_4 + 7H_2O$.		Blank Test.	Test With $FeSO_4 + 7H_2O$.
Sec.	Millivolt.		Sec.	Millivolt.		Sec.	Millivolts.	
60	.....	0.13	60	0.52	0.47	60	0.85	0.84
60	0.15	0.145	60	0.54	0.52	60	0.875	0.87
60	0.16	0.15	60	0.56	0.55	60	0.88	0.88
60	0.17	0.165	60	0.58	0.58	60	0.90	0.90
60	0.18	0.170	60	0.60	0.60	60	0.91	0.91
60	0.20	0.180	60	0.62	0.60	60	0.92	0.91
60	0.22	0.215	60	0.64	0.61	60	0.935	0.91
60	0.24	0.24	60	0.66	0.62	60	0.95	0.92
60	0.27	0.265	60	0.675	0.63	60	0.96	0.95
60	0.30	0.295	60	0.69	0.65	60	0.97	0.97
60	0.325	0.32	60	0.715	0.66	60	0.985	0.98
60	0.35	0.345	60	0.735	0.68	60	1.00	1.00
60	0.37	0.365	60	0.75	0.71	60	1.01	1.01
60	0.39	0.38	60	0.77	0.73	60	1.02	1.02
60	0.42	0.39	60	0.785	0.75	60	1.035	1.03
60	0.45	0.41	60	0.805	0.775	60	1.05	1.05
60	0.48	0.44	60	0.82	0.80	60	1.06	1.06
60	0.50	0.45	60	0.84	0.82	60	1.07	1.07

[13] *Metallurgie*, vol. ix., No. 6, p. 199 (Mar. 22, 1912).

[14] *Journal der russischen physikalisch-chemischen Gesellschaft*, vol. xli., p. 680 (1909).

[15] *Chemisches Centralblatt*, vol. lxxx., p. 1124 (1908).

Table II. gives the observations made with the dehydration of green vitriol, $FeSO_4 + 7H_2O$. Column 1 shows that a record was taken every 60 sec.; in column 2 are noted the observations made with the empty boat; and in column 3 those with the boat charged with 0.3 g. of sulphate. A graphical representation of the data is given in Fig. 2, in which the ordinate gives the millivolts observed and the abscissa the time in minutes. The upper smooth curve, full-drawn, represents the blank test. In the lower curve the dotted line is a copy of the curve of the blank test; the full-drawn line gives the observations made with the charged boat; the numerical data added

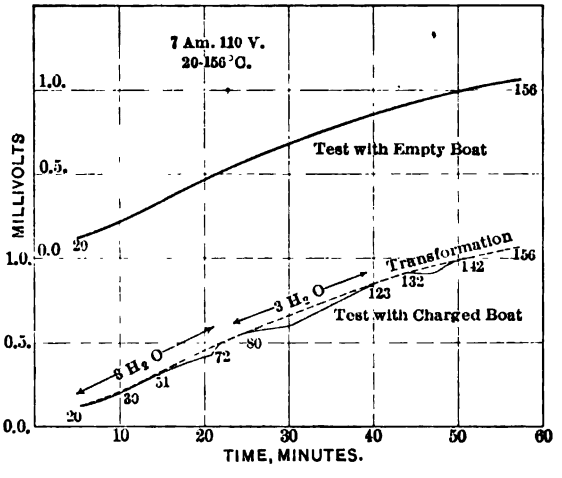


FIG. 2.—DEHYDRATION OF $FeSO_4 + 7H_2O$ IN AN OPEN TUBE WITH AIR-CURRENT.

to the full-drawn line represent the degrees centigrade scale corresponding to the readings on the millivoltmeter. Following the pair of curves, they coincide until the temperature of 20° C. has been reached, when a decided retardation takes place, which lasts until the temperature has risen to 72° C. Weighings of the boat show that in the interval between 20° and 72° C. 3 molecules of water have been expelled. The weight of the charged boat now remains constant between 72° and 80° C. Here a second retardation is noticed, which extends from 80° to 123° C., and corresponds to an additional loss of 3 molecules of water, proved by the difference in weight. Between 123° and 132° the line curves coincide; if not theoreti-

cally, they do it at least practically. In the interval between 132° and 142° there is no change in weight; the retardation indicates a transformation, which is shown by a change in the color of the salt from white to gray. The expulsion of the last molecule of water is accompanied by the formation of a basic ferric salt, discussed on page 534.

II. Plan of Investigation.

The original plan of the investigation was, to determine the dissociation-temperatures of anhydrous metallic sulphates, the formation of basic sulphates, the characters of the resulting products, and the speeds of reaction at different temperatures, in order to obtain data applicable to roasting-operations. As the start had to be made in most cases with the chemically pure hydrous sulphates, the original plan was extended to include the dehydration of the hydrous salts by heating in a current of air. This research, besides having a general interest, has the practical value of determining (1) the temperatures at which sulphates with given amounts of water of crystallization can separate from their solutions, and (2) the temperatures to which crystals may be warmed in artificial drying without losing some water of crystallization. Thus, in order to obtain crystals of blue vitriol of the composition $CuSO_4 + 5aq$, the crystallization may not begin before the solution has cooled down to below 28° C., as otherwise $CuSO_4 + 3aq$ would separate, because at 28° C., $CuSO_4 + 5aq$ gives up 2 molecules of water; for the same reason the drying of crystals of $CuSO_4 + aq$ may not be carried on at a temperature above 28° C.

The method of heating in a current of purified and dried air was used in the dehydration-tests for the same reasons as those in desulphatization.

Two other methods have been used by early investigators. Thus Debray[16] applied the vacuum method discussed on page 524; Müller-Erzbach,[17] what may be called the evaporation method. In the latter the vapor-tension of a saturated salt-solution and of water are compared by placing the two in same-size tubes in a desiccator charged with concentrated sulphuric acid. The loss of water in the two cases gives the ratio of

[16] *Comptes rendus de l'Académie des Sciences*, vol. lxvi., No. 4, p. 194 (1868).

[17] *Annalen der Physik und Chemie*, vol. xxiii., p. 617 (1884).

vapor-pressures. The ratio remains constant as long as the original salt solution remains unaltered, it changes as soon as a new hydrate begins to be formed. Thus sodium phosphate, $Na_2HPO_4 + 12aq$, loses at first 5 molecules of H_2O, then again 5, and lastly 2 molecules, proving thus the existence of three hydrous compounds.

With the exception of the sulphates of copper, calcium, and barium, all the sulphates of any metallurgical importance have been investigated, and the new results compared with the earlier. The three sulphates first mentioned have been accepted, as they were carried out under similar conditions by us and a former associate. The paper concludes with a general review of the subject and two final tables of data, which we consider reliable for dehydration and desulphatization.

III. Heating-, Measuring-, and Testing-Apparatus.

The furnace used for heating was an electric resistance tube-furnace similar to the well-known Heraeus furnace employed by Hofman and Mostowitsch [18] in their work upon calcium sulphate, but containing a few improvements. Figs. 3 and 4 give a longitudinal section and an end-view. The heating-tube, *A*, of unglazed Marquardt material, 1.968 in. in diameter and 11.81 in. long, is wound with platinum foil; it is contained in a fire-clay tube, *B*, 3.543 in. inner diameter, and centered in it by fire-clay rings, *C*, at the ends, leaving an air-space, *D*, 0.6299 in. wide. The fire-clay tube is surrounded by calcined magnesia, *E*, inclosed in a sheet-iron cylindrical casing, *F*, 10.63 in. in diameter, with one flanged movable end, *G*, secured by screw-bolts, *H*, and thumb-nuts, *I*; the bolts are flattened at one end and riveted to the casing. The narrow annular space between casing *F* and tube *B* is closed with strips of asbestos paper soaked in starch-paste. The clay rings *C* are held in place by two triangular pieces of sheet-iron, *J*, backed by asbestos board, *K*, and attached to casing *F* by three threaded bolts, *L*, and nuts *M*. The silica tube *N*, 0.7874 in. in diameter and 19.68 in. long, lies in the center of the heating-tube *A*; the small open annular spaces in rings *C* are closed with a clay-cement plaster. The tube holds the porcelain boat *O*, 1.575 in. long by 0.2756

[18] *Trans.*, xxxix., 631 (1908).

in. wide by 0.1968 in. deep, which contains the substance to be tested; both ends of tube N are closed with perforated rubber stoppers, P; at the opening Q is inserted a ⊥ glass-tube for introducing the thermo-electric pyrometer, R, with protection

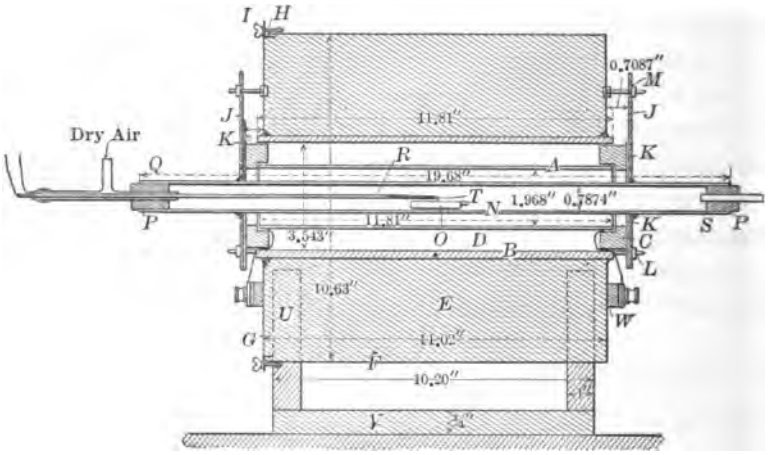


FIG. 3.—ELECTRIC RESISTANCE TUBE-FURNACE. LONGITUDINAL SECTION.

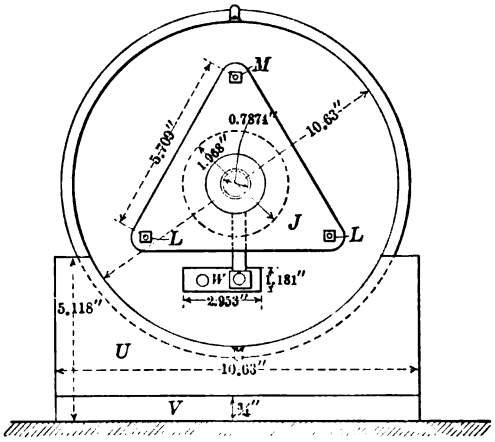


FIG. 4.—ELECTRIC RESISTANCE TUBE-FURNACE. END-ELEVATION.

silica tube, 0.1181 in. in diameter, isolating silica tube, 0.0295 in. in diameter, and 0.0098-in. walls, and admitting air under pressure; at the opening S is a glass tube to be connected with a railroad-tube holding the indicator. The hot junction, T, of

the couple is uncovered. The furnace rests in semi-circular recesses of two pieces of asbestos board, *U*, 1 in. thick, nailed to board, *V*, of the same material, 0.75 in. thick. Board *U* holds the binding-posts, *W*, for the electric connection.

During a test the uncovered thermo-couple was held above the substance; the electric measurements were made with a Siemens-Halske millivoltmeter having 180 decimillivolt divisions for 1,700° C.; the cold-junction was held in ice-water; the couple was standardized against the boiling-points of water, naphthaline and sulphur, and the freezing-points of water, aluminum, and the eutectic copper-cuprous oxide.

The current used was of 110 volts; it passed through the ammeter and two resistance-coils, one for rough and the other for close regulation. With all the resistance turned on, the current was of 2 amperes; with all turned off, the strength was 18 amperes, and the latter sufficed to attain 1,150° C.

It had been the intention to use a self-regulating rheostat, which was to consist of a non-conducting cylinder covered with resistance-wire, a gliding contact, and a clock to rotate the cylinder. This would have given straight lines instead of the curved ones shown in the records, but, unfortunately, the apparatus could not be made ready in time for this work. During our experiments an apparatus aiming at the same result, but of different construction, was described by Friedrich.[19]

In an experiment a certain number of amperes were made to pass through the winding of the furnace, and kept constant; records were taken every 60 sec.; the boat was removed from the furnace, transferred to a desiccator, and weighed; the substance was simply examined, or tested, as may have been found necessary.

Air under pressure was obtained by means of a Beutell apparatus connected with the water-service of the laboratory; the velocity was regulated by means of screw-clamps to furnish in the wash-bottles from 45 to 50 bubbles per minute. The air, before entering the furnace, was purified by passing it through wash-bottles containing a $\frac{1}{10}$ normal solution of potassium permanganate, concentrated potassium hydroxide, concentrated barium hydroxide, concentrated sulphuric acid, and a drying-

[19] *Stahl und Eisen*, vol. xxxi., No. 50, p. 2045 (Dec. 14, 1911).

tower, 15 in. high and 1.75 in. in diameter, charged with calcium chloride and phosphorus pentoxide.

The gases leaving the furnace were passed through a colorless solution of potassium iodate and starch[20] for the detection of SO_2, and through a solution of barium chloride, acidulated with hydrochloric acid, to test for SO_3.

In making dehydration-tests at temperatures not exceeding 120° C., the preliminary indications of the electric resistance furnace were followed up with a Reichert thermostat, provided with the gas-pressure regulator of Bausch & Lomb. This enabled us to hold temperatures for any length of time within from 1° to 2° C.

IV. Record of Results.

1. *Ferrous Sulphate*, $FeSO_4 + 7H_2O$.

This salt forms bluish-green monoclinic crystals, which weather on exposure to air, being superficially oxidized to a basic ferric sulphate. Besides the common salt, there exist[21] the compounds $FeSO_4 + 4H_2O$, $FeSO_4 + 1H_2O$, and $FeSO_4$. The salt $FeSO_4 + 7H_2O$ is said to lose[22] 6 molecules of H_2O at 115° and to retain the 7th at 280°. Heating with exclusion of air gives[23] white $FeSO_4$, while with access of air[24] the basic ferric sulphate, $Fe_2O_3.2SO_3$, is formed. Several tests have recently been made upon the behavior of $FeSO_4$ when heated in a current of air. Warlimont[25] found that dissociation began at 470° and was complete at 550°. Bradford[26] noted the first retardation at 540° and gives 550° as the temperature at which the salt is completely decomposed. Friedrich[27] noticed the first absorption of heat at 700° and gives the dissociation-temperature as 705°. Kothny[28] finally found that at 150° $FeSO_4$ begins to be oxidized without any expulsion of SO_2, that at

[20] Hofman, *Trans.*, xxxv., 817 (1904).

[21] Müller-Erzbach, *Annalen der Physik und Chemie*, vol. xxvi., p. 409 (1885).

[22] Graham, *Philosophical Magazine*, Third Series, vol. vi., No. 36, p. 421 (June, 1835).

[23] Brandes, *Schweigger's Journal*, vol. li., p. 438 (1827).

[24] Waltl, *Chemisches Repertorium für Pharmacie*, vol. xli., p. 428 (1902).

[25] *Metallurgie*, vol. vi., No. 4, p. 131 (Feb. 22, 1909).

[26] *Trans.*, xxxiii., 50 (1902).

[27] *Metallurgie*, vol. vii., No. 11, p. 330 (June 8, 1910).

[28] *Oesterreichisches Jahrbuch*, vol. lviii., p. 115 (1910).

480° it is completely converted into $Fe_2O_3 \cdot 2SO_3$, with a loss of only 4.2 per cent. of the S in the form of SO_2, and that at 530° the basic salt is completely dissociated into Fe_2O_3 and SO_3, but very slowly.

A chemical analysis of the salt used in the tests is given in Table III.

TABLE III.—*Composition of $FeSO_4 + 7H_2O$.*

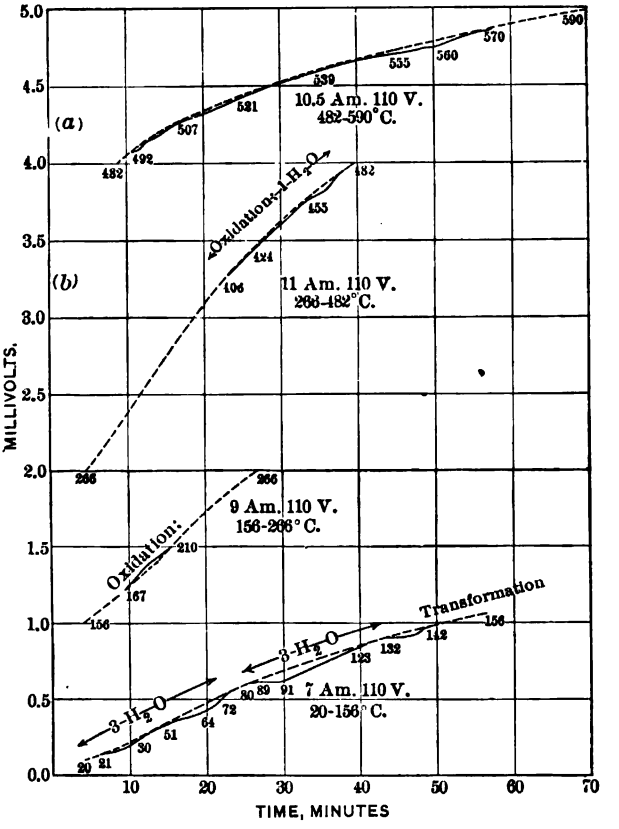
Component.	Chemical Analysis.	Calculated Analysis.
	Per Cent.	Per Cent.
SO_3	28.82	28.77
FeO	26.01	25.91
H_2O	45.17[a]	45.32
SO_3 : FeO	1.1080	1.1103

[a] By difference.

The iron was precipitated, after oxidation with nitric acid, from a boiling solution with ammonia, and weighed after ignition as Fe_2O_3. The sulphur was determined as $BaSO_4$, from a chloride solution, by precipitating with barium chloride.

The substance used in the test weighed 0.3 g., and the charged boat 4.1672 g. The bottom curve in Fig. 5 represents the heating from 20° to 156°. At 21° occurs the first retardation, which lasts until the temperature has risen to 30°; it begins again at 51°, and terminates at 72°. Parallel tests in the thermostat showed that heating to 25° for 8 hr. expelled 3 molecules of H_2O. At 80° there is a second deviation from the curve of the blank test, which disappears at 123°; again 3 molecules of H_2O have been drawn off. Test with the thermostat showed that heating to 120° for 25 hr. removed 3 molecules of H_2O. The absorption of heat lasting from 132° to 142° corresponds to a transformation; the substance, originally white, has become gray, and has not undergone any change in weight. The weight of the charge was 4.0568 g. after the temperature had reached 156°; further heating for 1.75 hr. at this temperature reduced it to 4.0547 g., and for an additional hour to 4.0527 g. This corresponds to a total reduction of 0.1145 g., which is equal to 6 molecules of H_2O. Between 156°

and 266° the curve for the charged boat rises above that for the blank test, on account of the heat set free by the oxidation of the ferrous salt. The boat, removed after 266° had been reached, weighed 4.0534 g.; exposing it for 0.5 hr. more to 266° increased the weight only to 4.0535 g. The substance, gray after the transformation, had become a greenish yellow; the formation of a basic salt had begun, but the 7th molecule of



FIGS. 5*a* AND 5*b*.—FERROUS SULPHATE, $FeSO_4$, + $7H_2O$.

H_2O had not yet been expelled. Between 266° and 482° no change occurs up to 406°, when there is a slight deviation for the curve of the charged boat from that of the blank test, reaching to 455°, when the absorption of heat becomes more marked; now the boat weighed 4.0484 g. Returning the boat to the furnace cooled to 475°, heating for 20 min. reduced the weight to 4.0466 g., for 30 min. more to 4.0418 g., and again

for 30 min. to 4.0420 g. The substance has become brownish-yellow. Two processes have taken place, the ferrous iron has been oxidized, and the 7th molecule of H_2O has been expelled. The total loss in weight was 0.1252 g., *i. e.*, 0.0107 g. less than called for by the 7 molecules of H_2O (= 0.1359 g.); the difference is due to the oxidation of the ferrous iron, accompanied by the formation of Kothny's $Fe_2O_3.2SO_3$.

The curve in Fig. 5*a* extending from 482° to 590° shows at 492° the first retardation, due to the beginning of the dissociation of the $Fe_2O_3.2SO_3$. The process proceeds very slowly and becomes marked only between 550° and 560°. This range of temperature can, therefore, be taken as the practical temperature for the decomposition of $FeSO_4$, which, however, is noticeable already at 492°.

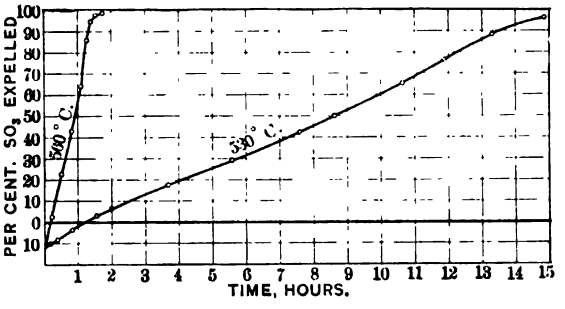


FIG. 6.—VELOCITY OF DECOMPOSITION OF $FeSO_4$ IN A CURRENT OF AIR AT 530° AND 560° C.

The velocity of decomposition at 530° and 560° C. is shown in Fig. 6.

There exist three hydrous salts, $FeSO_4 + 7H_2O$; $FeSO_4 + 4H_2O$; $FeSO_4 + 1H_2O$. The salt $FeSO_4 + 7H_2O$ gives up 3 molecules of H_2O at 21°, 3 more at 80°, undergoes a transformation at 132°, begins to be oxidized at 167°, but very slowly; the oxidation continues with the rise of temperature to 455°, when the oxidation to $Fe_2O_3.2SO_3$ is completed, and at the same time the last molecule of H_2O is expelled. The dissociation of $Fe_2O_3.2SO_3$ begins at 492° and is finished at from 555° to 560°.

2. *Bismuth Sulphate*, $Bi_2(SO_4)_3$.

The normal sulphate is a white powder which readily com-

bines with H_2O to form $2Bi_2(SO_4)_3 + 7H_2O$.[29] The decomposition of the anhydrous salt[30] begins between 405° and 418°; at a higher temperature $Bi_2O_3.SO_3$ is formed[31]; upon further increasing the heat the dissociation is complete,[32] but, according to Schmidt,[33] is accompanied by the volatilization of some Bi_2O_3. Abegg[34] doubts the existence of basic sulphates.

The normal salt used in the experiments contained 5 per cent. of H_2O, as shown in Table IV. It was treated with water and some ammonium sesquicarbonate was added, the resulting basic bismuth carbonate was filtered, dried, fused in a porcelain crucible with five times its weight of 98-per cent. potassium cyanide, and the resulting metal weighed. The sulphur was determined by precipitating with barium chloride from a chloride solution and weighing as $BaSO_4$. The results are given in Table IV.

TABLE IV.—*Composition of* $Bi_2(SO_4)_3$.

Component.	Chemical Analysis.	Calculated Analysis.
	Per Cent.	Per Cent.
SO_3	37.75	39.735
Bi_2O_3	57.25	60.265
H_2O	5.00[a]	none
$SO_3 : Bi_2O_3$	0.6595	0.6594

[a] By difference.

The substance tested weighed 0.3 g.; the weight of the charged boat was 5.0144 g. Heating to 170° reduced the weight to 4.9994 g. by the expulsion of 0.0150 g. of H_2O. No retardations were noted in the curve, Fig. 7*a*, while heating from 170° to 347°, and from 347° to 495°; the indicator remained uncolored, and the weight of the boat unchanged. The salt was light yellow when hot, but became again white when cold.

[29] Gmelin-Kraut, *Handbuch der anorganischen Chemie*, vol. iii., part ii., p. 981 (1908).

[30] Bailey, *Journal of the Chemical Society*, vol. li., p. 676 (1887).

[31] Heintz, *Annalen der Physik und Chemie*, vol. lxiii., p. 77 (1844).

[32] Bailey, *loc. cit.*

[33] *Berichte der deutschen chemischen Gesellschaft*, vol. xxvii., pt. 1, No. 2, p. 236 (1894).

[34] *Handbuch der anorganischen Chemie*, vol. iii., part ii., p. 665 (1907).

In the curve representing the rise of temperature from 495° to 666°, Fig. 7*b*, a retardation is seen at 570°, which is due to decomposition setting in; this progresses very slowly to 603°, then becomes more noticeable up to 639°, when occurs a second decided retardation, explained by the formation of a basic salt, yellow when hot, though less so than the normal salt, and white when cold. In order to ascertain the composition of the basic

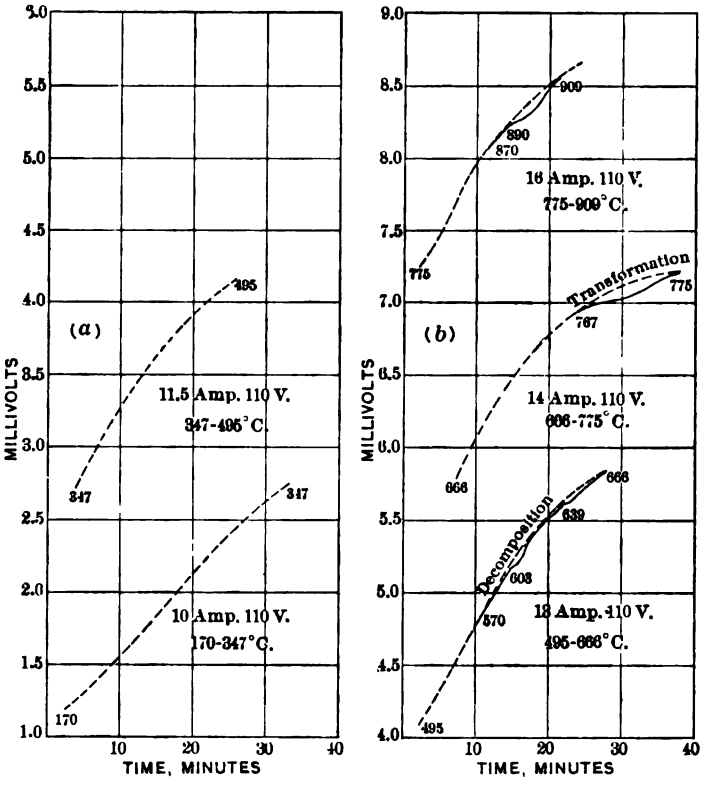


Fig. 7*a*. Fig. 7*b*.

FIGS. 7*a* AND 7*b*.—DECOMPOSITION OF $Bi_2(SO_4)_3$.

salt, 0.3 g. of substance was weighed out and brought to constant weight at 630°. The charged boat weighed 5.0144 g.; heating to 170° expelled the hygroscopic water and reduced the weight to 4.9994 g. Heating at 630° for five 20-min. periods gave the weights 4.9832, 4.9804, 4.9770, 4.9740, and 4.9744 g.; that is, a loss of 0.0250 g., referred to the weight of dry substance. This loss corresponds to one-fifth of the total

SO_3, and gives the basic salt the formula $5Bi_2O_3.4(SO_3)_3$. This is white, both hot and cold, and begins to form already at 570°. The curve giving temperatures ranging from 666° to 775° has a retardation, which begins at 767° and corresponds to a transformation. The salt has not changed either in weight or in color, but has shrunk greatly in volume. In heating from 775° to 909° there is noted an absorption of heat at 870°, accompanied by a coloration of the indicator, which shows the beginning of the dissociation of $5Bi_2O_3.4(SO_3)_3$; this is very decided at 890°, but not yet complete. At 922° (not shown) another retardation was seen, and this was due to fusion of the partly decomposed basic salt. Heating the fused salt to 900° for 4.5 hr. and to 1,020° for 1 hr. gave a diminution in weight of 0.0280 g., due in part to dissociation of the basic salt, in part to volatilization.

The normal anhydrous salt, $Bi_2(SO_4)_3$, begins to be converted at 570° into $5Bi_2O_3.4(SO_3)_3$, but the change progresses slowly; the basic salt undergoes a transformation at 767°, and a partial decomposition at 870°, which is not completed when the salt fuses at 922° and is further heated for a long time at 1,020° with partial volatilization.

3. *Aluminum Sulphate,* $Al_2(SO_4)_3 + 16H_2O$.

Pure alum crystallizes with 16 molecules of H_2O, forming a stable compound; the impure salt often crystallizes with 18 molecules of H_2O and is hygroscopic.[35] Upon heating, the hydrous salt is said to fuse in its water of crystallization and to be decomposed without forming a basic salt.[36] Pickering[37] found that the so-called basic salts are mixtures of Al_2O_3 and $Al_2(SO_4)_3$. Wöhler-Plüddemann-Wöhler[38] determined the dissociation-temperature of the anhydrous salt at 750°; Friedrich[39] noticed retardations at 100° and 350°, and fixed the decomposition at 770°.

The composition of the salt used in the experiments is given in Table V.

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[35] Mendelejeff, *Grundlagen der Chemie*, p. 751 (1891).
[36] Dammer, *Handbuch der anorganischen Chemie*, vol. iii., p. 98 (1893.)
[37] *Chemical News*, vol. xlv., pp. 121, 133 (Mar. 24, 31, 1882).
[38] *Berichte der deutschen chemischen Gesellschaft*, vol. xli., pt. 1, No. 4, p. 703 (1908).
[39] *Metallurgie*, vol. vii., No. 11, p. 333 (June 8, 1910).

TABLE V.—*Composition of* $Al_2(SO_4)_3 + 16H_2O$.

Component.	Chemical Analysis.	Calculated Analysis.
	Per Cent.	Per Cent.
SO_3	38.08	38.081
Al_2O_3	16.12	16.22
H_2O	45.80 [a]	45.699
$SO_3 : Al_2O_3$	2.3635	2.3479

[a] By difference.

The alumina was precipitated with ammonia and weighed as Al_2O_3; the sulphur was separated from a chloride solution with barium chloride and weighed as $BaSO_4$.

For the experiments 0.3 g. was taken; the charged boat weighed 4.1100 g. The bottom curve in Fig. 8*a* represents heatings from 15° to 130°. The first retardation was noticed at 51° and extended to 80°, the second lasted from 82° to 97°, the third from 97° to 105°, and the fourth from 109° to 130°. Each of the first three retardations is caused by an expulsion of 3 molecules of H_2O. Parallel tests in the thermostat showed that heating at 80° for 30 hr. eliminated 3 molecules of H_2O, at 87° for 6.5 hr. again 3 molecules, at 105° for 5.5 hr. 3 more molecules, and at 170° for 33 hr. again 3 molecules of H_2O. The long time required to remove the last 3 molecules finds a parallel in the fact that in the tube-furnace at 130° only part of these 3 molecules had been expelled, the time given being too short. The charged boat heated to 130° weighed 4.0230 g.; the loss in weight had been 0.0870 g. instead of 0.1027 g. In the second curve, covering from 130° to 276°, the temperature has risen to 180° before the 12th molecule was completely expelled. Three more molecules of H_2O are driven out between 180° and 263°. The boat, after being held for 30 min. at 263°, weighed 3.9938 g.; for an additional 60 min., 3.9866 g.; and for 30 min. more, 3.9860. g. It took 120 min. at 263° to expel the 15th molecule of H_2O, which shows the tenacity with which the water is held. The next two curves, covering 276° to 380° and 380° to 544°, Figs. 8*a* and 8*b*, show the elimination of the 16th and last molecule of H_2O, which starts at 316° and is finished at 442°. The weight of the

charged boat had been reduced to 3.9732 g., which corresponds to a loss of 0.1368 g., while the theoretical figure calls for 0.1363 g. The salt was snow-white. In the last curve, covering from 544° to 746°, the dissociation of the anhydrous salt is seen to begin at 590°; it progresses slowly, becomes marked at 639°, grows weaker, and is complete at 690°. The charged

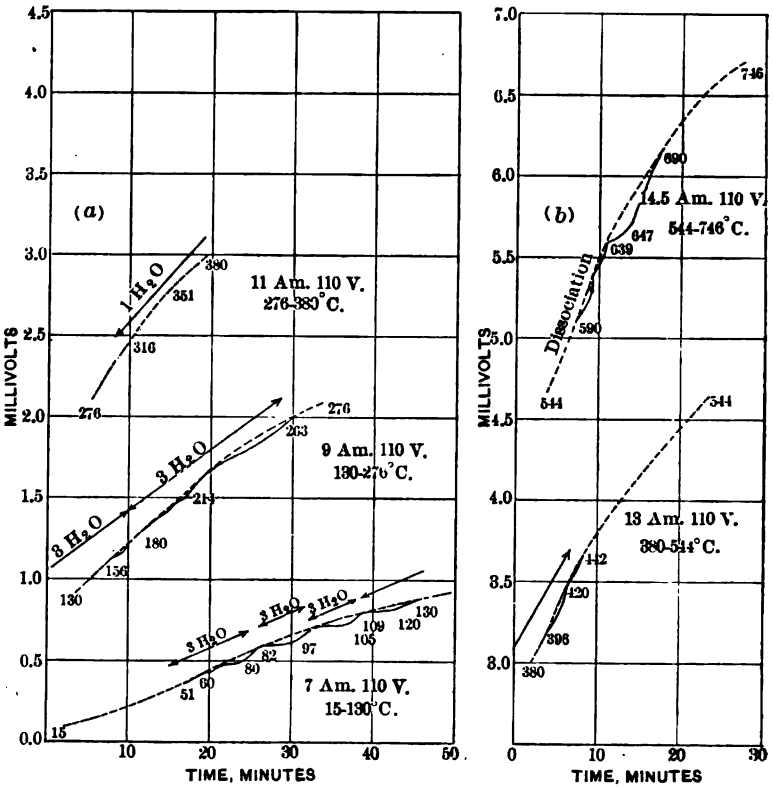

Fig. 8a. Fig. 8b.

FIGS. 8a AND 8b.—DEHYDRATION AND DESULPHATIZATION OF $Al_2(SO_4)_3 + 16H_2O$.

boat, removed at 746°, weighed 3.9545 g., which is equivalent to a loss of 0.0187 g. and corresponds to the SO_3 given off. The velocity of decomposition at 755° is represented graphically in Fig. 9.

There exist six hydrous salts: $Al_2(SO_4)_3 + 16H_2O$; $Al_2(SO_4)_3 + 13H_2O$; $Al_2(SO_4)_3 + 10H_2O$; $Al_2(SO_4)_3 + 7H_2O$; $Al_2(SO_4)_3 + 4H_2O$; $Al_2(SO_4)_3 + 1H_2O$.

The salt $Al_2(SO_4)_3 + 16H_2O$ gives off 3 molecules of H_2O at 51°, 3 at 82°, 3 at 97°, 3 at 109°, 3 at 180°, and the last one at 316°. The dissociation of the anhydrous salt begins at 590° and becomes marked at 639°.

4. *Lead Sulphate*, $PbSO_4$.

Anglesite forms orthorhombic crystals, the prepared sulphate a white crystalline powder.

In a general way it is known that this salt loses some of its SO_3 at elevated temperatures. Doeltz and Graumann[40] found that in heating in an air-current for 30-min. periods the salt

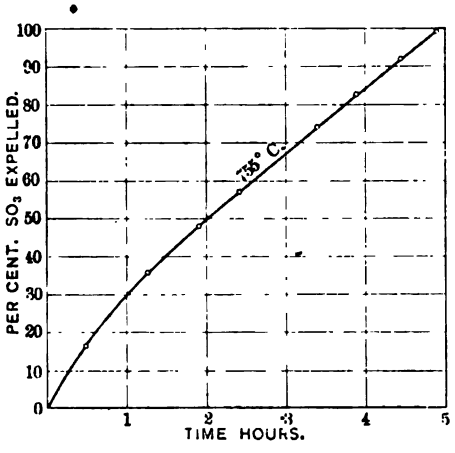

FIG. 9.—VELOCITY OF DECOMPOSITION IN A CURRENT OF AIR OF $Al_2(SO_4)_3$ AT 755° C.

suffered no change in weight up to 800°, that at 900° a loss of from 0.32 to 0.42 per cent. took place, that at 1,000° much SO_3 was expelled, causing in 1.5 hr. a reduction in weight of 14.3 per cent. Ramsay and Eumorfopoulos[41] determined the melting-point of $PbSO_4$ as 937°, which according to the research of Schenck and Rassbach (see below) must have been that of the partly decomposed salt. Boussingault[42] claims to have decomposed the salt completely. Berthier[43] prepared the compound $2PbO.SO_3$ by fusion of $PbSO_4$ and PbO; he says it is a light-yellow substance when hot, and white when cold. Schenck and

[40] *Metallurgie*, vol. iii., No. 13, p. 441 (July 8, 1906).
[41] *Philosophical Magazine*, Fifth Series, vol. xli., No. 251, p. 360 (Apr., 1896).
[42] *Annales de Chimie et de Physique*, Fourth Series, vol. xii., p. 246 (1867).
[43] *Op. cit.*, Second Series, vol. xliii., p. 287 (1830).

Rassbach [44] have traced the freezing-point curve of the series $PbSO_4$—PbO. They found that the melting-point of $PbSO_4$ lay above 1,100°, and that at 950° a retardation occurred, which was caused by a partial decomposition. Their curve shows the existence of two stable basic sulphates: $2PbO.SO_3$, with melting-point of 960°, and $3PbO.SO_3$, with melting-point of 951°; a third basic salt, $4PbO.SO_3$, is stable only below 880°, as above this temperature it splits into $3PbO.SO_3$ and PbO. They noted also two transformations, $PbSO_4$ at 850°, and $3PbO.SO_3$ at 450°.

An analysis of the lead sulphate used in the experiments gave the results shown in Table VI.

TABLE VI.—*Composition of $PbSO_4$.*

Component.	Chemical Analysis.	Calculated Analysis.
	Per Cent.	Per Cent.
SO_3	26.13	26.43
PbO	73.45	73.57
SO_3 : PbO	0.3557	0.3592

For the experiment there was weighed out 0.3 g. of substance; the charged boat weighed 4.1690 g. Heating to 181° reduced the weight to 4.1682 g. by the expulsion of water. The curves, Fig. 10*a*, covering 181° to 297° and 297° to 553°, show slight deviations for the charged boat between 195° and 238°, and between 345° and 418°; both are accompanied by bluing of the indicator, showing that some SO_2 is set free. The loss in weight caused by the first retardation was 0.0012 g., that by the second 0.0080 g., or 2.66 per cent. The white color of the original substance had changed to gray. Heating for an additional 20 min. at 553° caused no further loss in weight; it did, however, restore the original white color. In the temperature-range between 553° and 750°, Fig. 10*b*, retardations are noticeable at 637° and 705°; both are again accompanied by a slight setting-free of SO_2. The boat heated to 750° weighed 4.1566 g.; heating for one more hour at this

[44] *Berichte der deutschen chemischen Gesellschaft*, vol. xli., pt. 2, No. 13, p. 2917 (1908).

temperature reduced the weight to 4.1560 g. The total loss in weight endured so far was 0.0130 g., which corresponds to one-sixth of the SO_3 of 0.3 g. of substance. In order to verify these observations, again 0.3 g. of substance was taken (weight of charged boat was 4.3402 g.) and heated to 750°. After 4 or 5 min. the bluing of the indicator showed that SO_2 was

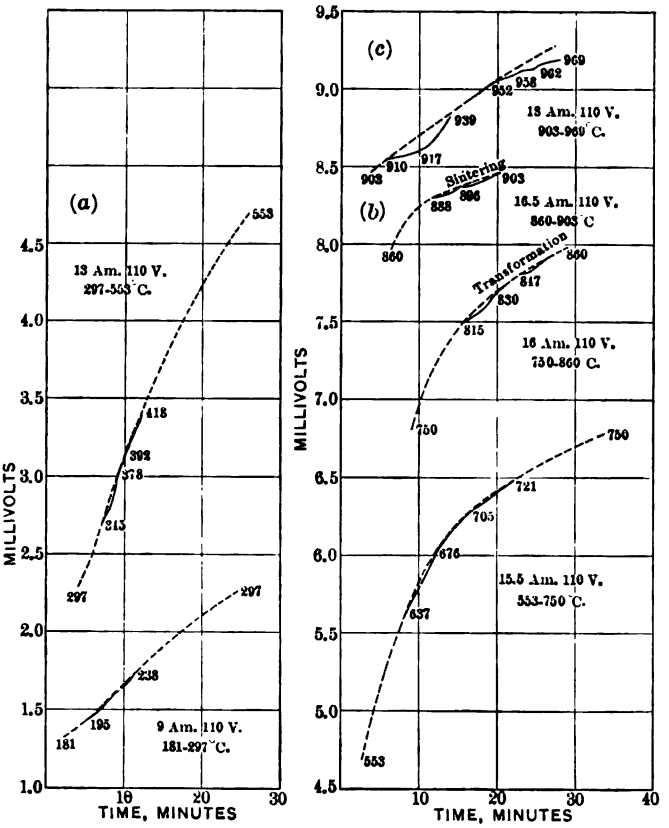

Fig. 10*a*. Figs. 10*b* and 10*c*.

FIGS. 10*a*, 10*b*, AND 10*c*.—DESULPHATIZATION OF $PbSO_4$.

being liberated. After 35 min. at 750° the weight of the charged boat was 4.3283 g., after 30 min. more 4.3276 g., after an additional 20 min. 4.3276 g.; that is, no more loss took place. The total loss was equal to one-sixth of the SO_3 present, hence the formula for the newly-found basic salt is $6PbO.5SO_3$. Between 750° and 860° there is a small retarda-

tion at 815°, accompanied by the evolution of some SO_2, and a large one at 847°, the temperature remaining constant there for 1.5 min. The charged boat, taken out at 860°, weighed 4.3267 g., the slight loss being caused by the expulsion of some SO_2; there had, however, taken place a large shrinkage in the volume of the substance, showing that a transformation of the salt $6PbO.5SO_3$ had occurred, which is in agreement with the results of Schenck and Rassbach. Heating the charged boat for an additional 30 min. at 860° gave a weight of 4.3265 g., or a loss of only 0.0002 g. The curve 860°–903° shows two retardations, at 888° and 896°; the first is caused by the decomposition of $6PbO.5SO_3$, proved by the coloration of the indicator and by the loss in weight, the boat at 903° weighing 4.3236 g.; the second by the partial sintering of the salt. The last curve, Fig. 10*c*, giving the results obtained between 903° and 969°, shows marked differences between the curves for the charged and empty boats. At 910° the fusion of the salt begins and is completed at 939°; dissociation with evolution of SO_3 begins at 952°; at 958° and 962° retardations were noticed, which may correspond to Schenck and Rassbach's compounds $3PbO.SO_3$ and $2\ PbO.SO_3$. In order to test the expulsion of SO_2, again 0.3 g. of $PbSO_4$ was weighed out, the charged boat weighing 4.3402 g. Heating to 950° reduced the weight to 4.3265 g., holding at 950° for 8 hr. 10 min. diminished it to 4.2700 g., and for 20 min. more to 4.2698. Raising the temperature to 1,115° and holding the boat at this heat for 30 min. lowered the weight to 4.2596 g., holding it for 120 min. more gave 4.2580 g. The loss in weight of 0.0102 g., experienced between the last weight taken at 950° and the first at 1,115°, viz., 4.2698 and 4.2596, exceeded already the theoretically possible loss of SO_3 (*i. e.*, 0.0792 g.), and thus proved that the decomposition of the basic salt $2PbO.SO_3$ is accompanied by the volatilization of PbO.

The results show that the decomposition of $PbSO_4$ begins already at 195° but progresses very slowly, with the formation of the basic salt $6PbO.5SO_3$ at 705°; that this salt undergoes a transformation at 847°, begins to be decomposed at 888°, to sinter at 896°, and to fuse at 910°; rapid dissociation begins at 952°, and is accompanied by volatilization of PbO; retarda-

tions at 958° and 962° indicate an acceleration of the decomposition, which, however, is not complete.

5. *Cupric Sulphate*, $CuSO_4 + 5H_2O$.

Blue vitriol forms sky-blue triclinic crystals. The expulsion of its water of crystallization has been studied by Lescœur,[45] Andreae,[46] Müller-Erzbach,[47] and Wanjukow.[48] They are all in accord as to the existence of the following salts: $CuSO_4 + 5H_2O$; $CuSO_4 - 3H_2O$; $CuSO_4 + 1H_2O$; $CuSO_4$.

According to Magnier de la Source,[49] $CuSO_4 + 5H_2O$ is changed in a current of air of 25° to 30° into sky-blue $CuSO_4 + 3H_2O$. This has been re-investigated by Wanjukow, who fixed the range of conversion between 27° and 30°; he found also that the pale-blue $CuSO_4 + 1H_2O$ was formed between 93° and 99°. Graham[50] says that blue vitriol is completely dehydrated between 221° and 243°; Wanjukow found that the complete dehydration began already at 155° and was completed at 233°.

The decomposition of $CuSO_4$ by heating was probably first studied by Rousche,[51] who found that at a dark red the basic orange-colored salt $2CuO.SO_3$ was formed. In more recent times Bradford[52] found that in an air-current $CuSO_4$ was decomposed to $2CuO.SO_3$ at 653°, and the latter at 702° into CuO and SO_3.

Warlimont[53] ascertained that by heating for 3 hr. the decomposition of $CuSO_4$ to CuO and SO_3 began between 600° and 610°, and increased at 630° to 4.8 per cent., at 640° to 11.3 per cent., and at 650° to 100 per cent. Landis[54] noticed the first traces of SO_2 at 400°; a rapid evolution of gas took place at 690°; this diminished with rise of temperature until 900°

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[45] *Comptes rendus de l'Académie des Sciences*, vol. cii., No. 25, p. 1466 (1886).

[46] *Zeitschrift für physikalische Chemie*, vol. vii., p. 241 (1891).

[47] *Annalen der Physik und Chemie*, vol. xxvi., p. 409 (1885); *Zeitschrift für physikalische Chemie*, vol. xix, p. 144 (1896).

[48] *Journal der russischen physikalisch-chemischen Gesellschaft*, vol. xli., p. 710 (1909).

[49] *Comptes rendus de l'Académie des Sciences*, vol. lxxxiii., No. 20, p. 899 (1876).

[50] *Philosophical Magazine*, Third Series, vol. vi., No. 36, p. 418 (June, 1835).

[51] *Zeitschrift für Chemie und Pharmazie*, vol. iii., p. 462 (1860).

[52] *Trans.*, xxxiii., 59 (1902).

[53] *Metallurgie*, vol. vi., No. 4, p. 132 (Feb. 22, 1909).

[54] *Metallurgical and Chemical Engineering*, vol. vii., No 1, p. 327 (Jan., 1910).

was reached, when the dissociation was complete. Friedrich[55] found by his method that $2CuO.SO_3$ was formed at 740°, that this began to be decomposed between 840° and 850°, and that all sulphurous gas was expelled at 900°. Wanjukow[56] noticed retardations at 341°, 380°, 503°, and 621°, all accompanied by evolutions of SO_2 and the formation of increasing small amounts of the basic salt $8CuO.3SO_3$. Thus, at 341°, there was decomposed 0.51 per cent. of the $CuSO_4$; at 380°, 1.09 per cent.; at 503°, 2.38 per cent.; and at 621°, 5.87 per cent. In treating with water this $CuSO_4$, thus partly converted into $8CuO.3SO_3$, the $CuSO_4$ went quickly into solution, while the basic salt remained unchanged for 30 min.; boiling for 20 min. caused it to take up some water and form a pale-blue hydrous salt, which, however, was not dissolved. Heating the partly decomposed $CuSO_4$ to 653° caused the formation of the orange-colored basic salt $2CuO.SO_3$, which was slow up to 670°, when it progressed more rapidly. The expelled gas contained both SO_3 and SO_2. This salt did not change during an exposure to dry air for 4 months; treating it with water at ordinary temperature had no effect for 5 or 6 min., then gradually the salt took up some water and became bluish, but was not dissolved. Heating $2CuO.SO_3$, the first retardation was noticed at 704°, the decomposition to CuO became pronounced at 736°, and continued until the temperature had risen to 751°. At 787° the residue contained 0.1108 per cent. of SO_3, at 802° and 805° only a trace.

There exist three hydrous salts: $CuSO_4 + 5H_2O$; $CuSO_4 + 3H_2O$; and $CuSO_4 + 1H_2O$. The salt $CuSO_4 + 5H_2O$, upon heating in a current of dry air, is converted between 27° and 30° into $CuSO_4 + 3H_2O$, the latter at from 93° to 99° into $CuSO_4 + 1H_2O$, and the last at 155° into $CuSO_4$. The anhydrous $CuSO_4$ is converted at 341° into $8CuO.3SO_3$ to the limit of 5.87 per cent.; at 653° $2CuO.SO_3$ begins to be formed; at 704° the final dissociation into CuO is started; it becomes pronounced at 736°.

6. *Manganous Sulphate*, $MnSO_4 + 5H_2O$.

This salt forms peach-blossom monoclinic crystals, which

[55] *Metallurgie*, vol. vii., No. 11, p. 327 (June 8, 1910).

[56] *Journal der russischen physikalisch-chemischen Gesellschaft*, vol. xli., p. 688 (1909).

slowly weather on exposure to air. Müller-Erzbach[57] recognizes three hydrous salts, with 5, 2, and 1 molecules of H_2O. Heating $MnSO_4 + 5H_2O$ to 150° expels 4 molecules of H_2O (Petterson[58]), and to 280° causes the removal of all the H_2O (Thorpe-Watts[59]); Graham[60] gives from 187° to 200° as the critical temperature. According to Friedrich,[61] the expulsion of H_2O begins at 280°; he found that at 700° the anhydrous salt fused, and that at 1,030° dissociation took place.

The salt used in the experiments was somewhat weathered, as shown by the water-content in Table VII.

TABLE VII.—*Composition of $MnSO_4 + 5H_2O$.*

Component.	Chemical Analysis.	Calculated Analysis.
	Per Cent.	Per Cent.
SO_3	35.50	33.22
MnO	31.45	29.43
H_2O	33.05[a]	37.35
SO_3 : MnO	1.1287	1.1287

[a] By difference.

The manganese was determined by the Volhard method; the sulphate was precipitated from a chloride solution with barium chloride and weighed as $BaSO_4$.

For the experiment 0.3 g. of substance was taken; the charged boat weighed 4.1652 g. In Fig. 11*a*, the bottom curve gives the changes taking place between 22° and 115°. The first retardation is noticed at 25°; it extends to 60° and is accompanied by a loss of 3 molecules of H_2O. During the second retardation, lasting from 60° to 87°, another molecule of H_2O is expelled. Parallel tests in the thermostat showed that up to 25° no water was expelled, that heating to 30° for 44 hr. eliminated 3 molecules of H_2O, and to 100° for 25 hr. part of the 4th molecule, but that the temperature had to be raised to

[57] *Annalen der Physik und Chemie*, vol. xxvi., p. 421 (1885).

[58] Gmelin-Kraut, *Handbuch der anorganischen Chemie*, vol. ii., part 2, p. 282 (1908).

[59] *Journal of the Chemical Society*, vol. xxxvii., p. 113 (1880).

[60] *Philosophical Magazine*, Third Series, vol. vi., No. 36, p. 420 (June, 1835).

[61] *Metallurgie*, vol. vii., No. 11, p. 329 (June 8, 1910).

120° to remove it completely. At 115° the charged boat weighed 4.0950 g., showing that a loss of 0.0702 g. had been experienced, while the expulsion of 4 molecules of H_2O raised the figure to 0.0741 g. The curve representing the range 115°–241° shows two retardations, at 148° and 187°, and the joining of the curves at 204°. The expulsion of the 5th molecule of H_2O begins at 152°. At 241° the charged boat weighed 4.0912 g.; holding it at this temperature for 1 hr. reduced the weight to 4.0716 g., and for 2 hr. more to 4.0676 g. The total loss was 0.0976 g.; the weight of 5 molecules of H_2O corresponds to 0.0991 g. The curves covering the ranges 241°–364°, and 364°–603°, Fig. 11*b*, show no thermal changes; the weight of the charged boat remained constant at 4.0676 g.

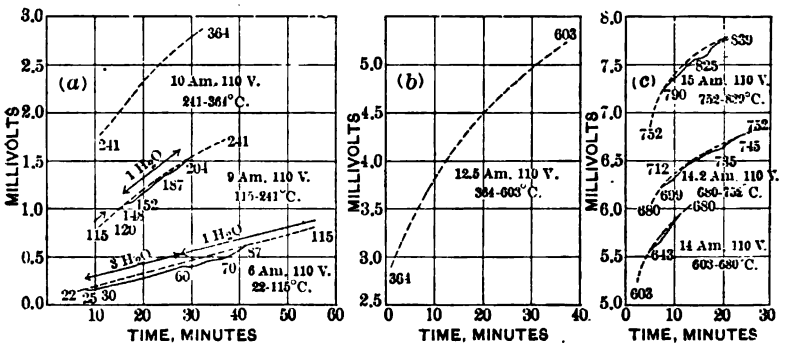

Fig. 11*a*. Fig. 11*b*. Fig. 11*c*.

FIGS. 11*a*, 11*b*, AND 11*c*.—DEHYDRATION AND DESULPHATIZATION OF $MnSO_4 + 5H_2O$.

The bottom curve of Fig. 11*c*, representing the heating from 603°–680°, shows an absorption of heat beginning with 643°, caused by a sintering of the edges of the crystals. Between 680° and 752° there is a decided retardation at 699°, which extends to 745°, and is caused by the dissociation of the anhydrous salt, the indicator showing the presence of SO_2 by turning blue. The salt upon examination was found to be not fused, and to have retained its pulverulent character. In order to fix more definitely the dissociation-temperature of $MnSO_4$, a new sample of 0.3 g. of substance was taken; the charged boat weighed 4.3310 g. Heating to 570° expelled all the water and reduced the weight to 4.2390 g. Holding it for

30 min. at 570° gave the weight 4.2384 g., the indicator showed the presence of SO_2; 30 min. at 700° did not change the weight, but the indicator began to show a bluish tinge; 30 min. at 710° reduced the weight to 4.2378 g. and decidedly colored the indicator, 30 min. at 730° gave 4.2372 g., 30 min. at 815° gave 4.2264 g., and the substance in the boat began to blacken. The last curve, representing heatings from 752° to 839°, has a clear retardation at 790°, which is due to a further loss of SO_2.

The velocities of decomposition of $MnSO_4$ at 770° and 840° are shown in Fig. 12.

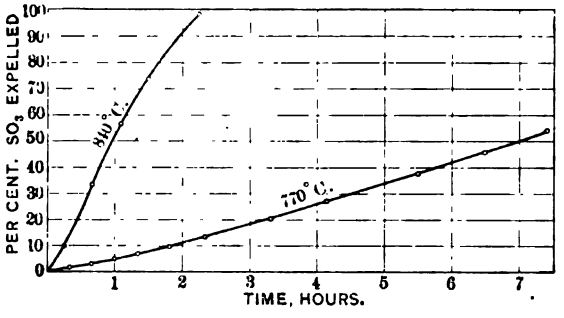

FIG. 12.—VELOCITY OF DECOMPOSITION OF $MnSO_4$ IN A CURRENT OF AIR AT 770° AND 840° C.

There exist three hydrous sulphates: $MnSO_4 + 5H_2O$, $MnSO_4 + 2H_2O$, $MnSO_4 + 1H_2O$. The salt $MnSO_4 + 5H_2O$ drops 3 molecules of H_2O at 25°, the 4th at 60°, and the 5th at 152°. Dissociation begins at 699° and becomes decided at 790°.

7. *Zinc Sulphate*, $ZnSO_4 + 7H_2O$.

Zinc vitriol forms orthorhombic white lustrous crystals, which disintegrate upon exposure to air for any length of time and give off some water.[62] Besides $ZnSO_4 + 7H_2O$, Müller-Erzbach[63] distinguishes the following sulphates: $ZnSO_4 + 2H_2O$, $ZnSO_4 + 1H_2O$, and $ZnSO_4$. Upon heating, zinc vitriol melts in its combined water. According to Wiedemann,[64] heating $ZnSO_4 + 7H_2O$

[62] Péchard, *Annales de Chimie et de Physique*, Sixth Series, vol. xxviii., p. 415 (1893).

[63] *Annalen der Physik und Chemie*, vol. xxvi., p. 409 (1885).

[64] *Annalen der Physik und Chemie*, vol. xvii., p. 561 (1882).

to 69° converts it into $ZnSO_4 + 6H_2O$; Graham[65] dehydrated it completely at 204°; Hofman[66] found 263° the lowest temperature at which all the water was expelled excepting 0.24 per cent., which could be driven off only at 528°. With a 2-g. sample it took from 30 to 40 hr. to complete a test. Mostowitsch[67] dehydrated the salt at 400° in from 1 to 2 hr. Friedrich[68] found that all the water was expelled at 300°. Heating the anhydrous salt in a current of dry air, Hofman[69] found that with the elimination of the last of the water at 528°, some SO_2 was set free for a short time and then stopped; raising the temperature caused more SO_2 to be liberated in different stages unto complete dissociation, the lowest temperature for which he found to be 739°. It takes a long time to accomplish this result. Mostowitsch[70] decomposed completely 0.6104 g. $ZnSO_4$ at 750° in 27 hr.; it took him only 2 hr. to do the same at 850°. Doeltz-Graumann[71] noted at 700° the first decomposition of $ZnSO_4$. Landis[72] noticed the first traces of SO_2 at 730°; the decomposition was rapid at 760° and apparently complete, as no SO_2 was detected afterward even when the temperature was raised to 980°. Friedrich[73] found his first retardation at 740°, which was caused by a transformation; the second was noticed at 840°, and was due to the formation of the basic salt $3ZnO.2SO_3$; the third, at 935°, was accompanied by the dissociation of the basic salt into ZnO and SO_3. Mostowitsch[74] could not detect a trace of either SO_2 or SO_3 up to 560°; above 560° the test-tube showed the presence of sulphur, and the substance began to lose in weight; the loss at 600° was only 0.40 per cent. of the SO_3 present. The decomposition of about 0.4 g. of $ZnSO_4$ at temperatures ranging from 650° to 900°, heating for 1-hr. periods, is shown in Fig. 13. The curve shows that at 850° the dissociation of $ZnSO_4$ is completed in 1 hr., which is a prac-

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[65] *Philosophical Magazine*, Third Series, vol. vi., No. 35, p. 333 (May, 1835).
[66] *Trans.*, xxxv., 817 (1904).
[67] *Metallurgie*, vol. viii., No. 24, p. 766 (Dec. 22, 1911).
[68] *Metallurgie*, vol. vii., No. 11, p. 328 (June 8, 1910).
[69] *Loc. cit.*
[70] *Metallurgie*, vol. viii., No. 24, p. 768 (Dec. 22, 1911).
[71] *Metallurgie*, vol. iii., No. 13, p. 445 (July 8, 1906).
[72] *Metallurgical and Chemical Engineering*, vol. viii., No. 1, p. 22 (Jan., 1910).
[73] *Metallurgie*, vol. vii., No. 11, p. 328 (June 8, 1910).
[74] *Metallurgie*, vol. viii., No. 24, p. 763 (Dec. 22, 1911).

tical figure. It brings out another point, and this is that up to 700° the expulsion of SO_3 is quick, and above 700° uniformly slow. The diagrams of Hofman[75] tell a similar story, viz., that the dissociation progresses rapidly until one-third of the SO_3 has been expelled, and then more slowly. Both diagrams indicate the presence of an intermediary basic salt, the existence of which has been shown by Friedrich, and verified in the present tests, but at a much lower temperature.

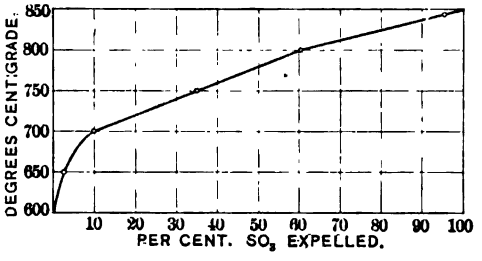


FIG. 13.—DEGREE OF DECOMPOSITION OF $ZnSO_4$ IN 1-HR. PERIODS.

The result of the chemical analysis of the salt used in the experiments is given in Table VIII.

TABLE VIII.—*Composition of* $ZnSO_4 + 7H_2O$.

Component.	Chemical Analysis.	Calculated Analysis.
	Per Cent.	Per Cent.
SO_3	27.90	27.84
ZnO	28.39	28.32
H_2O	43.71 [a]	43.84
SO_3 : ZnO	0.9821	0.9830

[a] By difference.

The zinc was precipitated with sodium carbonate and weighed as ZnO; the S was separated with barium chloride from a chloride solution and weighed as $BaSO_4$.

For the experiment 0.3 g. of substance was taken; boat and substance weighed 4.0912 g. It was heated in the thermostat up to 25° for 2.5 hr., whereby the weight was reduced to 4.0725 g.; the loss of 0.0187 g. corresponds to 1 molecule of

[75] *Trans.*, XXXV., 822 (1904).

H_2O, as indicated in the bottom curve of Fig. 14*a*. Between 28° and 110° a retardation is seen to start at 28° and to extend to 100°; from 40° upward to 69° the curve for the charged boat runs nearly parallel with that for the empty boat, and recedes again suddenly. The charged boat heated to 110° weighed 3.9980 g., showing a loss in weight of 0.0745 g., which corresponds to 4 molecules of H_2O (= 0.0749 g.). Parallel tests with the thermostat showed that heating the original substance to 35° for 24 hr. eliminated 5 molecules of H_2O. Be-

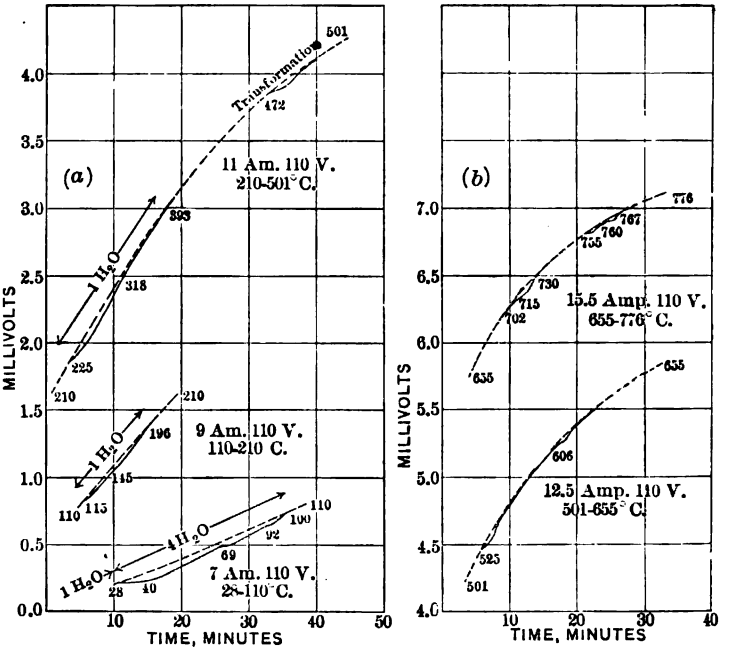


Fig. 14*a*. Fig. 14*b*.

FIGS. 14*a* AND 14*b*.—DEHYDRATION AND DESULPHATIZATION OF $ZnSO_4 + 7H_2O$.

tween 110° and 210° a retardation is seen between 115° and 196°. The boat at 210° weighed 3.9800 g.; the loss of 0.0180 g. is equal to the weight of 1 molecule of H_2O (= 0.0187 g.). Heating for an additional 30 min. produced no further change in weight. Between 210° and 501° two absorptions of heat are evident, which extend from 225° to 393°, and from 472° to 501°. The first is due to an almost complete expulsion of the 7th molecule of H_2O, the second to a transformation accompanied

by a slight dissociation, proved by the momentary bluing of the test-solution, and an expulsion of some water. The weight of the boat at 501° was 3.9625 g.; the total loss in weight of 0.1287 g. shows that all the water (= 0.1315 g.) has not yet been driven off. The substance was white, but with a grayish tinge. Between 501° and 655°, Fig. 14*b*, the first retardation, at 525°, is accompanied by a decided emission of SO_2 and an expulsion of the last of the H_2O; the decomposition progresses very slowly up to 606°, when the advance is more quick, but on the whole still very slow. At 655° the boat weighed 3.9605 g. In the last curve, from 655° to 776°, there is noticed an absorption of heat between 702° and 730° accompanied by the splitting-off of both SO_2 and SO_3, which, as shown below, is equal to one-third of the SO_3 present, and corresponds therefore to the formation of the basic salt $3ZnO.2SO_3$. The second thermal effect occurs at 755°, and is due to the decomposition of $3ZnO.2SO_3$. The boat at 776° weighed 3.9536 g. In order to test the formation of this basic salt, 0.3 g. of original substance, boat and salt weighing 3.9776 g., was dehydrated and heated to 730°, the termination of the first retardation, and held there for 4 hr. The salt free from H_2O ought to weigh 3.9776 — 0.1315 = 3.8461 g.; the weight after 4 hr. at 730° was 3.8186 g.; the difference of 0.0275 g. corresponds to the theoretical value 0.0280 g. for SO_3, if the basic salt $3ZnO.2SO_3$ has been formed, hence we are justified in accepting the formula.

There exist four hydrous salts: $ZnSO_4 + 7H_2O$; $ZnSO_4 + 6H_2O$; $ZnSO_4 + 2H_2O$; $ZnSO_4 + 1H_2O$. The salt $ZnSO_4 + 7H_2O$, heated to 25°, gives up 1 molecule of H_2O, at 28° there are split off 4 molecules, at 115° the 6th, and at 225° the 7th molecule; a transformation occurs at 472°. Heating the anhydrous salt to 702° causes decomposition with the formation of $3ZnO.2SO_3$, which in its turn is dissociated at 755°.

8. *Nickel Sulphate*, $NiSO_4 + 7H_2O$.

Morenosite occurs in the form of apple-green thin prisms; the artificial nickel vitriol forms emerald-green crystals, which weather on exposure to air, losing 1 molecule of H_2O (Dobroserdoff).[76] According to Müller-Erzbach,[77] there exist

[76] Moissan, *Traité de Chimie Minérale*, vol. iv., p. 279 (1905).

[77] *Annalen der Physik und Chemie*, vol. xxvi., p. 409 (1885).

three hydrous sulphates: $NiSO_4 + 7H_2O$; $NiSO_4 + 6H_2O$; $NiSO_4 + 1H_2O$. Hauer[78] states that heating $NiSO_4 + 7H_2O$ to 100° causes 4 molecules of H_2O to be expelled; Graham[79] found that at 103° there were driven off 6 molecules of H_2O, and that the 7th followed at 279°, while Pierre[80] places the complete dehydration at 200° to 250°. Friedrich[81] found that at 500° the vitriol was completely freed from water.

In heating the anhydrous normal salt, a basic salt is said to be formed.[82] Warlimont[83] found that the dissociation of $NiSO_4$ began at 600°, heating for 3 hr. at 625° caused 8.4 per cent. of the SO_4 to be expelled, an additional 3 hr. at 650° gave 33 per cent., and 3 hr. more at 670° gave 88 per cent. Friedrich[84] gives 840° as the beginning of the dissociation.

The nickel vitriol used in the experiments gave, upon chemical analysis, the results recorded in Table IX.

TABLE IX.—*Composition of $NiSO_4 + 7H_2O$.*

Component.	Chemical Analysis.	Calculated Analysis.
	Per Cent.	Per Cent.
SO_3	29.96	29.98
NiO	28.00	28.01
H_2O	42.04[a]	42.01
SO_3 : NiO	1.0700	1.0703

[a] By difference.

The nickel was determined electrolytically in an ammoniacal solution; the sulphur was precipated with barium chloride from a chloride solution, and weighed as $BaSO_4$.

For the experiment there was taken 1.0774 g.; the charged boat weighed 8.2456 g. Heating to 32° caused no change in weight. Between 32° and 60°, Fig. 15, retardations were ob-

[78] *Sitzungsberichte der Akademie d. Wissenschaften, Wien*, vol. xxxix., p. 305 (1860).

[79] Gmelin-Kraut, *Handbuch der anorganischen Chemie*, vol. v., part ii., p. 72 (1902).

[80] *Annales de Chimie et de Physique*, Third Series, vol. xvi., p. 252 (1846).

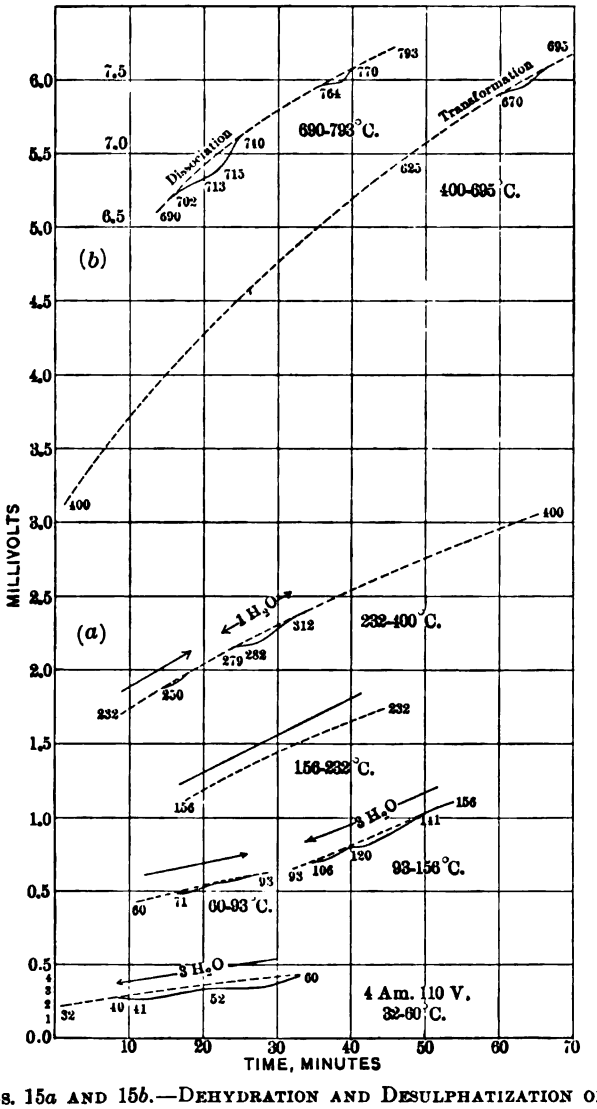
[81] *Metallurgie*, vol. vii., No. 11, p. 327 (June 8, 1910).

[82] Tupputi, in Dammer, *Handbuch der anorganischen Chemie*, vol. iii., p. 507 (1893).

[83] *Metallurgie*, vol. vi., No. 4, p. 132 (Feb. 22, 1909).

[84] *Metallurgie*, vol. vii., No. 11, p. 327 (June 8, 1910).

served at 40° and 52°; the boat heated to 60° weighed 8.2350 g., showing a loss of some water; holding it for 7 hr. at 60° reduced the weight to 8.0850 g., which would correspond to



FIGS. 15*a* AND 15*b*.—DEHYDRATION AND DESULPHATIZATION OF $NiSO_4 + 7H_2O$.

about 2.5 molecules of H_2O. Between 60° and 93° there is seen a retardation at 71°; the boat at 93° weighed 8.0818 g., the salt had become a pale sky-blue; holding it for 135 min. at 93°

reduced the weight to 8.0790 g., showing that 3 molecules of H_2O had been set free; the color begins to change to orange-yellow. Parallel tests in the thermostat showed that heating to 40° needed as much as 110 hr. to expel these 3 molecules of H_2O. In the temperature-range between 93° and 156°, water begins to be removed at 106°, the reaction weakens and becomes again strong at 120°. Heated to 156° the boat weighed 8.0732 g.; holding it for 4.25 hr. at 130° reduced this weight to 7.8740 g., which would correspond to a total expulsion of a little over 5½ molecules of H_2O, or, the second 3 molecules of H_2O had not been entirely set free. Between 156° and 232° the curve of the charged boat about coincides with that of the empty boat. In heating from 232[b] to 400°, there occurs an absorption of heat at 250°, explained by the driving-off of the rest of the 6th molecule of H_2O. The retardation at 279° is caused by the almost entire removal of the 7th molecule of H_2O; the color of the substance changes to an orange-yellow. The boat brought to 400° and weighing 7.800 g. showed that 0.641 per cent. of 1 molecule of H_2O was still held by the substance. Between 400° and 695° a decided retardation takes place at 670°, unaccompanied by any change in weight; it is due to a transformation which changes the orange-colored salt to a grayish white. In the temperature-range between 690° and 793° there is a thermal effect at 702°, which reaches a maximum at 715°, disappears at 740°, has a duration of 8 min., and shows the dissociation of the salt. With the dissociation takes place the expulsion of the remaining small amount of the 7th molecule of H_2O. In order to verify the dissociation-temperature, a new sample of 0.3 g. of vitriol was taken (weight of charged boat, 5.0082 g.) and heated; at 690° the first SO_3 was set free, but the loss in weight amounted only to 0.1264 g. (7 molecules of H_2O = 0.1261 g.); at 740° it rose to 0.1392 g. While the dissociation started at 690°, it became decided at 702°. Between 740° and 793° there was noticed the last retardation, extending from 764° to 770°, which shows an acceleration of the dissociation.

The velocity of decomposition of $NiSO_4$ at 710° and 770° is shown in Fig. 16.

There exist three hydrous salts: $NiSO_4 + 7H_2O$; $NiSO_4 + 4H_2O$; $NiSO_4 + 1H_2O$. The salt $NiSO_4 + 7H_2O$ begins to lose

3 molecules of H_2O at 40°, but parts with them very slowly; the same is the case with the second 3 molecules of H_2O, which begin to be set free at 106°; the expulsion of the 7th molecule of H_2O begins at 279°, but is not completed until the dissociation of the salt into NiO and SO_3 begins, which takes place at 702°.

9. *Cobalt Sulphate*, $CoSO_4 + 7H_2O$.

The mineral bieberite forms flesh- to rose-red monoclinic crystals; those of the prepared cobalt vitriol are similar. They are stable if separated from a neutral solution, unstable if the solution was acid.[85] Müller-Erzbach[86] distinguishes three hydrates: $CoSO_4 + 7H_2O$; $CoSO_4 + 6H_2O$; $CoSO_4 + 1H_2O$. Friedrich[87] found that all the water was expelled at 420°, and

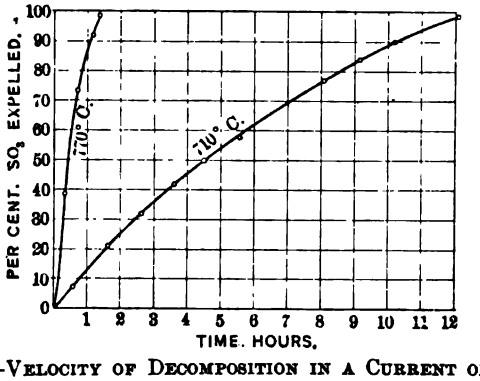


FIG. 16.—VELOCITY OF DECOMPOSITION IN A CURRENT OF AIR OF $NiSO_4$ AT 710° AND 770° C.

that dissociation began at 880°. Hartley[88] says that the anhydrous salt is unchanged at 300°, becomes lilac-colored between 500° and a red-heat, but regains its original rose-color upon cooling. According to Barth,[89] the anhydrous salt is stable at 600°, heating for 3 hr. at 613° caused a decomposition of 3.4 per cent. of the salt, and for 2 hr. at 735° the complete dissociation.

A chemical analysis of the salt used in the experiments gave the results shown in Table X.

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[85] Baubigny and Péchard, *Comptes rendus de l'Académie des Sciences*, vol. cxv., No. 3, p. 171 (1892).

[86] *Annalen der Physik und Chemie*, vol. xxvi., p. 409 (1885).

[87] *Metallurgie*, vol. vii., No. 11, p. 326 (June 8, 1910).

[88] *Journal of the Chemical Society*, vol. lxxxiii., p. 404 (1903).

[89] *Metallurgie*, vol. ix., No. 6, p. 202 (Mar. 22, 1912).

TABLE X.—*Composition of* $CoSO_4 + 7H_2O$.

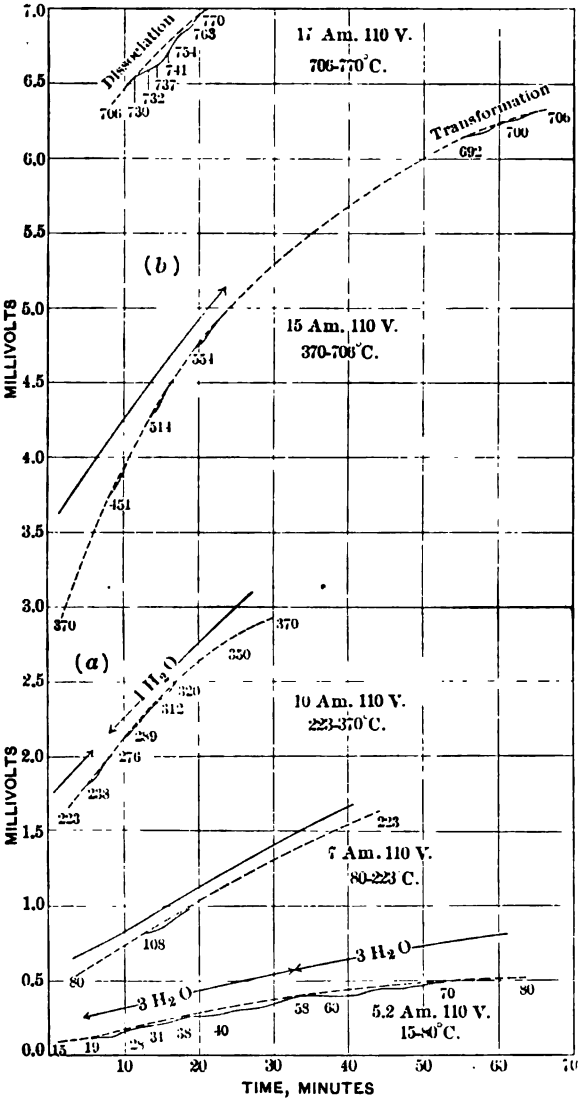
Component.	Chemical Analysis.	Calculated Analysis.
	Per Cent.	Per Cent.
SO_3	28.54	28.53
CoO	26.57	26.59
H_2O	44.89 [a]	44.88
$SO_3 : CoO$	1.0741	1.0728

[a] By difference.

The cobalt was determined electrolytically from an ammoniacal solution; the sulphur was precipitated with barium chloride from a chloride solution and weighed as $BaSO_4$.

The weight of the substance used for experiment was 0.3995 g.; the charged boat weighed 6.7860 g. The bottom curve, Fig. 17, representing the heating from 15° to 80°, shows two leading retardations, at 19° and at 58°. At 19° water begins to be separated, the expulsion becomes weaker at 28° and 38°, gains in force up to 40°, diminishes, and finally stops at 58°. Parallel tests in the thermostat showed that heating for 30 hr. at 36° caused the separation of 3 molecules of H_2O. At 58° begins the splitting-off of an additional 3 molecules of H_2O. Heated to 70° the charged boat weighed 6.6748 g., showing a loss of 0.1112 g. of H_2O, which would correspond to a total of 4⅓ molecules. Heating at 80° for 1 hr. increased the loss in weight; the boat weighed 6.6450 g., showing a decrease of 0.0298 g.; continuing the heating at 80° for 45 min. reduced the weight only very little, viz., 6.6444 g. The salt is homogeneous and purple-colored. In the temperature-range between 80° and 223° there is a thermic effect accompanied by a slight further reduction in weight, the boat, heated at 223°, weighing 6.6388 g.; holding it at 223° for an additional 30 min. produced no further expulsion of water. Between 223° and 370° the first absorption of heat is seen at 238°, which is due to the removal of the remaining water of the 6th molecule. At 276° the last molecule begins to be set free and is apparently completely liberated at 320°, when the boat weighed 6.6106 gr., showing that there had separated altogether 0.1754 g., which is less than 0.1793 g., the weight of 7 mole-

cules of H_2O. Heating to 340° caused the weight of the boat to fall to 6.6082 g., still not enough to form 1 molecule of H_2O. The salt was blue when hot, purple to rose-color when cold.



FIGS. 17*a* AND 17*b*.—DEHYDRATION AND DESULPHATIZATION OF $CoSO_4 + 7H_2O$.

Raising the temperature from 370° to 706°, Fig. 17*b*, causes three minor retardations, at 451°, 514° and 554°, which are caused by the removal of the rest of the 7th molecule of H_2O.

The weight of the boat at 706° was 6.6072 g., showing that 0.1788 g. had been expelled; the theoretical value is 0.1792 g. The heat-absorption seen at 692° is due to a transformation, as there is no change in weight nor any reaction with the indicator. In the last curve, covering the range from 706° to 770°, there is a strong thermal effect at 732°, accompanied by the setting-free of SO_2. The bluing of the indicator stopped at 737°, but started again at 754°. At 770° the boat weighed 6.6054 g., equal to a loss in weight of 0.1806 g. As the 7 molecules of H_2O of the substance weigh 0.1793 g., the difference between the two numbers shows that SO_2 has been expelled. In order to verify the temperature of expulsion of the last molecule of H_2O, and of the dissociation of the anhydrous salt, 0.3754 g. of fresh vitriol was taken; the charged boat

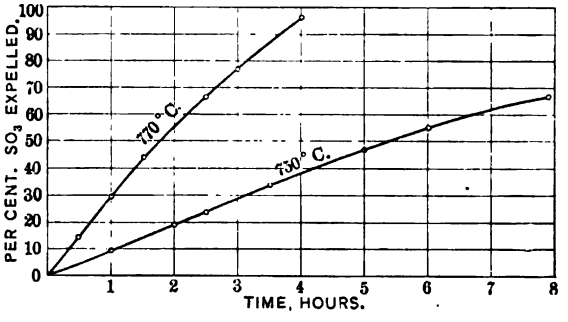


FIG. 18.—VELOCITY OF DECOMPOSITION OF $CoSO_4$ IN A CURRENT OF AIR AT 750° AND 770° C.

weighed 6.8072 g. Heating for 1 hr. at 650° caused the weight to be reduced to 6.6389 g.; the difference, 0.1683 g., corresponds to the theoretical weight of 7 molecules of H_2O, 0.1689 g., showing that at 650° the dehydration was complete. Heating to 690° and 700° for 1 hr. caused no change in weight nor any effect on the indicator; between 717° and 720° the indicator becomes blue after 15 min.; after holding 1 additional hour at this temperature, the loss in weight was 0.0030 gr.; 1 hr. at 725° caused a loss of 0.0128 g. It is therefore correct to say that the dissociation begins at 720°.

The velocity of decomposition at 750° and 770° is shown in Fig. 18.

There exist three hydrous salts: $CoSO_4 + 7H_2O$; $CoSO_4 +$

$4H_2O$; $CoSO_4 + 1H_2O$. The salt $CoSO_4 + 7H_2O$ begins to lose 3 molecules of H_2O at 19°, 3 more are expelled at 58°, and the last one is set free at 276°. The anhydrous salt undergoes a transformation at 692°, and is dissociated at 720°.

10. *Cadmium Sulphate*, $3CdSO_4 + 8H_2O$.

The formula of Richter[90] for hydrous cadmium sulphate crystallizing out at ordinary temperature has been used in the heading. The crystals readily disintegrate when exposed to air. Abegg[91] doubts the existence of any other hydrous salt, excepting $CdSO_4 + 1H_2O$; the existence of this has been proved by Lescœur[92]. According to Frémy,[93] the salt is dehydrated at a dark red. Mendelejeff[94] states that $CdSO_4$ is less readily decomposed by heat than $ZnSO_4$.

A chemical analysis of the substance used in the experiments gave the results shown in Table XI.

TABLE XI.—*Composition of* $CdSO_4 + 8H_2O$.

Component.	Chemical Analysis.	Calculated Analysis.
	Per Cent.	Per Cent.
SO_3	31.11	31.17
CdO	50.20	50.11
H_2O	18.69 [a]	18.72
SO_3 : CdO	0.6197	0.6223

[a] By difference.

The cadmium was determined electrolytically from a potassium cyanide solution according to the method of Beilstein and Jawein; the sulphur was precipitated from a chloride solution with barium chloride and weighed as $BaSO_4$.

For the experiment 0.3 g. of substance was used; the charged boat weighed 5.0078 g. The bottom curve, Fig. 19*a*, covering the temperature-range from 30° to 110°, shows that water was split off already at 30°, and that the thermic effect ceased

[90] *Lehrbuch der anorganischen Chemie*, bearbeitet von Klinger, Bonn, p. 386 (1906).
[91] *Handbuch der anorganischen Chemie*, vol. ii., part 2, p. 469 (1905.)
[92] *Annales de Chimie et de Physique*, Seventh Series, vol. iv., p. 222 (1895).
[93] *Encyclopédie Chimique*, vol. iii., p. 285 (1900).
[94] *Grundlagen der Chemie*, p. 714 (1891).

at 39°; the second separation of water began at 41°, and lasted until 90° was reached. Heated to 110° the boat weighed 4.9730° g.; further heating for 1 hr. did not cause any change in weight. The loss in water at 110° was, therefore, 0.0348 g., which corresponds to an expulsion of $1\frac{2}{3}$ molecules of H_2O. Parallel tests in the thermostat for 14 hr. at 37° showed that $\frac{2}{3}$ molecule of H_2O had been driven off, and for 15 hr. at 60° that 1 molecule of H_2O had been separated. In the range between 110° and 260° a retardation occurs at 170°, which extends to 220°. As the charged boat heated to 220° weighed 4.9518 g., the loss in weight corresponded to 1 molecule of H_2O.

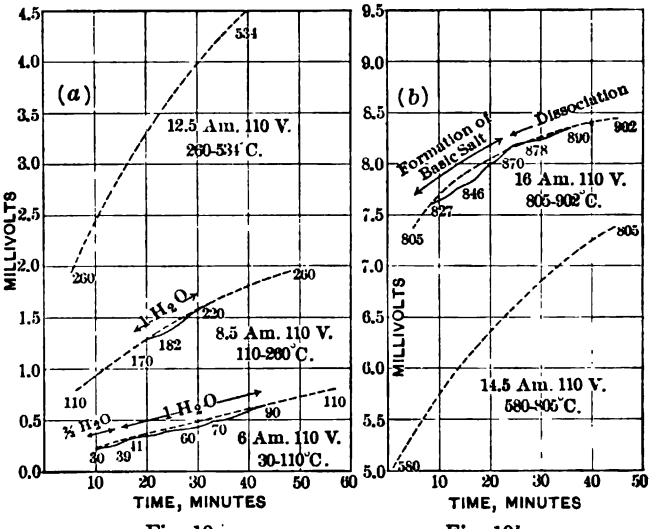


Fig. 19*a*. Fig. 19*b*.

FIGS. 19*a* AND 19*b*.—DEHYDRATION AND DESULPHATIZATION OF $CdSO_4 + 8H_2O$.

Between 260° and 580° (in Fig. 19*a* the curve is drawn only to 534°), and 580° and 805°, Fig. 19*b*, no thermal effects were noticed, but between 805° and 902° are seen two retardations; the first, lasting from 827° to 870°, is due to the formation of a basic salt; the second, extending from 878° to 890°, is caused by the dissociation of the basic salt into CdO and SO_3. The boat heated to 902° weighed 4.9460 g.; the loss in weight was 0.0058 g. Heating at 902° for 9 hr. reduced the weight to 4.8600 g.; there had been expelled 0.0918 g.; the theoretical value for SO_3 is 0.0935 g. The salt was white when the pro-

longed heating at 902° began, became yellow when hot, but turned again white upon cooling. After heating 4 hr. black specks were noticed in the white salt; these increased until at the end of the 9th hr. the substance became black. In order to verify the formation of the basic salt, another experiment with 0.3 g. of substance was made. The charged boat, weighing 3.6150 g., was heated to 850°, and held there for 12 hr.; its weight was reduced to 3.4840 g., which corresponds, after deducting the 0.0561 g. of H_2O expelled, to a loss of 0.0749 g. of SO_3, that is, four-fifths of the total SO_3. The formula for the basic salt is therefore $5CdO.SO_3$. Heating for two additional hours did not change the weight of the white salt. Raising the temperature to 880°, that is, above the starting-point of the

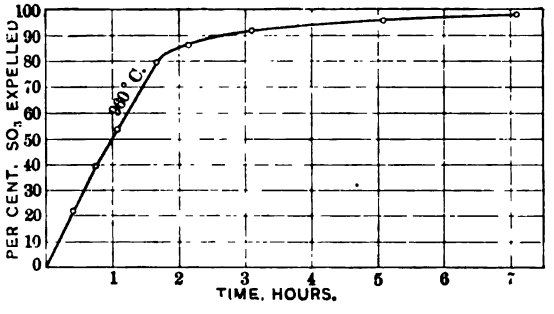


FIG. 20.—VELOCITY OF DECOMPOSITION IN A CURRENT OF DRY AIR OF $CdSO_4$ AT 960° C.

second retardation in the curve, causes expulsion of SO_3, and a change of the white color of the basic salt into the black of the oxide. The velocity of decomposition of $CdSO_4$ at 960° is shown in Fig. 20.

There exist three hydrous salts: $3CdSO_4 + 8H_2O$; $CdSO_4 + 2H_2O$; $CdSO_4 + \frac{1}{6}H_2O$. Heating $3CdSO_4 + 8H_2O$ to 30° causes two-thirds of 1 molecule of H_2O to be expelled, at 41° 1 molecule of H_2O separates off, and at 170° another. The anhydrous sulphate is decomposed at 827° to form the basic salt $5CdO.SO_3$, and this is finally dissociated into CdO and SO_3 at 878°.

11. *Magnesium Sulphate, $MgSO_4 + 7H_2O$.*

The natural as well as the artificial epsom salt forms white, translucent, orthorhombic crystals, which weather in dry, but

remain unaltered in moist, air (Abegg).[95] Warming causes them to melt in their water of crystallization. Heating to 50° gives a transition form of $MgSO_4 + 6H_2O$ (Van der Heide).[96] According to Graham,[97] heating to 100° furnishes crystals of $MgSO_4$ with a little more than 2 molecules of H_2O; at the same temperature in vacuo, in the presence of H_2SO_4, the slight excess of H_2O above 2 molecules is completely removed. He found that the 6th molecule of H_2O was expelled at 132°, and that the 7th was retained even at 237°. Bailey[98] says that the anhydrous salt is stable at 450°; Le Chatelier[99] determined its dissociation-temperature as 1,160°.

The composition of the salt used in the experiments is given in Table XII.

TABLE XII.—*Composition of $MgSO_4 + 7H_2O$.*

Component.	Chemical Analysis.	Calculated Analysis.
	Per Cent.	Per Cent.
SO_3	33.30	33.70
MgO	16.76	16.72
H_2O	49.94[a]	50.08
SO_3 : MgO	1.9318	1.9258

[a] By difference.

The magnesia was precipitated in the cold from an ammoniacal solution with sodium phosphate, ignited, and weighed as $Mg_2P_2O_7$; the sulphur was precipitated from a chloride solution with barium chloride and weighed as $BaSO_4$.

For the experiment there was taken 0.3 g.; the charged boat weighed 3.9773 g. In the bottom curve, Fig. 21*a*, extending from 17° to 142°, the first retardation occurs at 19°; it is slight, but is caused by the splitting-off of 1 molecule of H_2O. A parallel test in the thermostat showed that heating 0.3 g. of substance at 19° for 8.5 hr. expelled 1 molecule of H_2O. The second retardation, extending from 38° to 105°, is much

[95] *Handbuch der anorganischen Chemie*, vol. ii., part 2, p. 58 (1905).
[96] *Zeitschrift für physikalische Chemie*, vol. xii., p. 417 (1893).
[97] *Philosophical Magazine*, Third Series, vol. vi., No. 36, p. 421 (June, 1835).
[98] *Journal of the Chemical Society*, vol. li., p. 676 (1887).
[99] *Bulletin de la Société Chimique*, vol. xlvii., p. 300 (1887).

more decided than the first; it corresponds to the driving-off of 4 molecules of H_2O. Heating 0.3 g. in the thermostat at 38° for 43.5 hr. removed a total of 5 molecules of H_2O. At 112° is a third retardation, caused by the loss of 1 molecule of H_2O. The charged boat heated to 142° weighed 3.8638 g.; holding it for 30, 25, and 30 min. at this temperature decreased the weights to 3.8486, 3.8483, and 3.8483 g., showing that 0.1290 g. had been lost; the theoretical figure for loss of H_2O is 0.1284 g. Heating in the thermostat for 3 hr. at 120° expelled 6 molecules of H_2O. In the second curve, extending

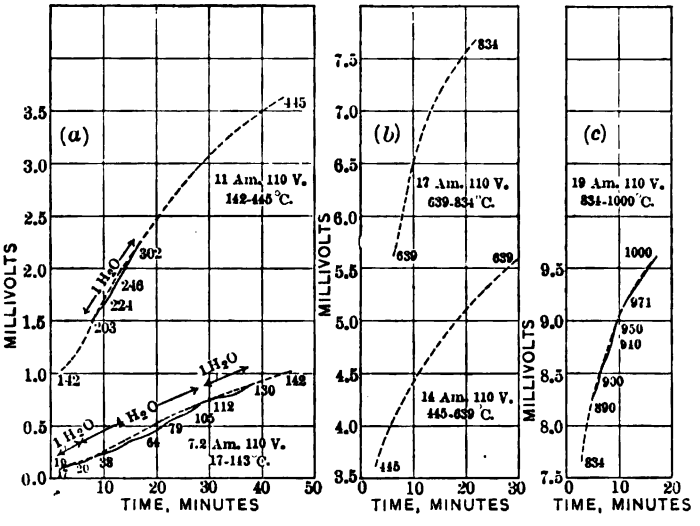


Fig. 21a. Fig. 21b. Fig. 21c.

FIGS. 21a, 21b, AND 21c.—DEHYDRATION AND DESULPHATIZATION OF $MgSO_4 + 7H_2O$.

from 142° to 445°, there occurs an absorption of heat at 203°, which lasts until the temperature has risen to 302°. The charged boat heated to 445° weighed 3.8275 g., corresponding to a total loss of 0.1509 g.; the weight of 7 molecules of H_2O in 0.3 g. of substance is 0.1489 g.; the salt, therefore, is completely dehydrated; it is white, with slight brownish shade. Raising the temperature from 445° to 639° and from 639° to 834° shows no thermal effects in the curves of Fig. 21b, the salt remains white. The desulphatization of the anhydrous salt is shown in Fig. 21c. The curve extends from 834° to 1,000°. At 890° there is seen a small deflection of the curve of the charged from that

of the empty boat; it increases from 890° to 940°, decreases, and gains again in magnitude at 950°. The indicator becomes colored at 900°. The dissociation begins at 890° and becomes decided at 971°.

Fig. 22 shows the velocity of decomposition of the anhydrous salt at 1,000°.

There exist four hydrous salts: $MgSO_4 + 7H_2O$; $MgSO_4 + 6H_2O$; $MgSO_4 + 2H_2O$; $MgSO_4 + 1H_2O$. Upon heating

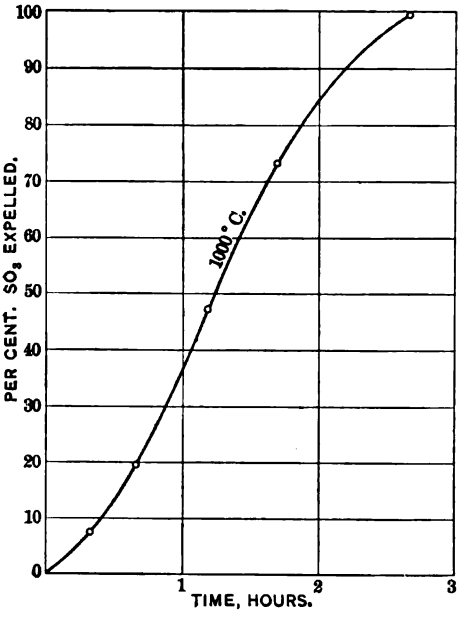


FIG. 22.—VELOCITY OF DECOMPOSITION IN A CURRENT OF AIR OF $MgSO_4$ AT 1,000° C.

$MgSO_4 + 7H_2O$ to 19° there is set free 1 molecule of H_2O, at 38° 4 molecules are liberated, the 6th molecule is split off at 112°, and the salt is dehydrated at 203°. Heating the anhydrous salt to 890° causes dissociation to begin.

12. *Silver Sulphate*, Ag_2SO_4.

This salt forms small white lustrous crystals. It is known to fuse when heated and to be decomposed at a temperature that lies higher than its melting-point. The only determinations of these temperatures are those by Bradford [100] and Fried-

[100] *Trans.*, xxxiii., 50 (1902).

rich.[101] Bradford determined the freezing-point at 655° and the dissociation at 1,095°, although he noticed losses in weight already at 900°, which increased with the temperature and the duration of the heating. Friedrich observed a transformation at 410°, when the white salt turned yellow; determined the melting-point at 660° and the dissociation-temperature at 1,085°.

A chemical analysis of the salt used in the experiments gave the results shown in Table XIII.

TABLE XIII.—*Composition of* Ag_2SO_4.

Component.	Chemical Analysis.	Calculated Analysis.
	Per Cent.	Per Cent.
SO_3	25.71	25.67
Ag_2O	74.21	74.33
$SO_3 : Ag_2O$	0.3464	0.3453

The silver was precipitated with dilute hydrochloric acid and weighed as AgCl, the sulphur with barium chloride from a chloride solution and weighed as $BaSO_4$.

In the experiments 0.3 g. of substance was used; the charged boat weighed 4.0924 g. Heating for 1.5 hr. to 260° gave a loss in weight of 0.0007 g.; that is, a trace of moisture was expelled. The lower curve, Fig. 23*a*, covers the range of temperature from 268° to 445°; at 417° is a decided retardation, caused by a transformation, as the weight remained unchanged (4.0917 g.) and the indicator was unaffected, but the color of the salt changed from white to a grayish yellow. In the upper curve, Fig. 23*a*, a thermal effect is noticed at 656°; the weight has not changed (4.0917 g.), but the salt has fused. Between 690° and 820°, Fig. 23*b*, there is only a slight thermal effect, which begins at 752° and extends to 917°; the weight is reduced very little, and the indicator shows that SO_3 is liberated in very small amounts. At 917° there is a strong absorption of heat; the full-drawn line runs horizontally for 3 min. and then bends upward; the indicator shows quickly the presence of much SO_3. When the first evolution of gas has ceased, it will appear again

[101] *Metallurgie*, vol. vii., No. 11, p. 330 (June 8, 1910).

when the temperature has risen and last for about 4 min.; it continues to do this until the dissociation is complete. The weight of the charged boat decreased at 917° from 4.0917 to 4.0778 g.; heating for three 30-min. periods at 940° reduced it to 4.0466, 4.0188, and 3.9910 g., when the salt had been complely converted into metallic silver.

The results show that a transformation takes place at 417°, fusion at 656°, and dissociation into Ag_2 and $SO_2 + O$ begins at 752° and becomes decided at 917°.

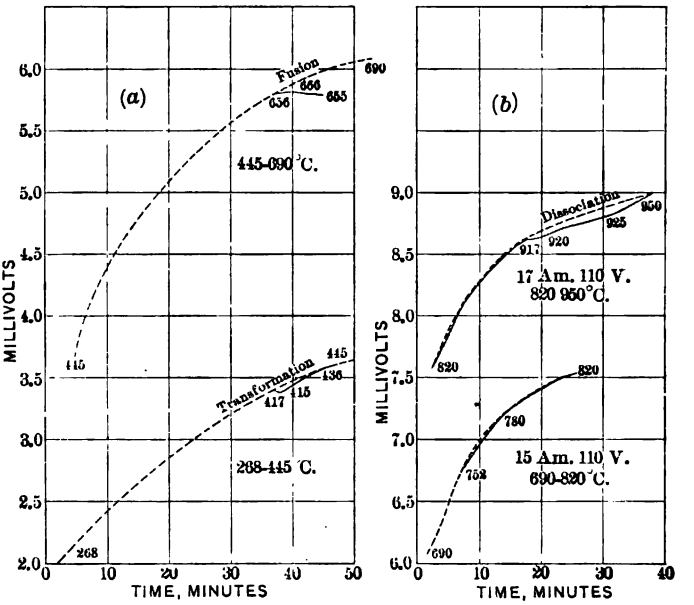


Fig. 23*a*. Fig. 23*b*.

FIGS. 23*a* AND 23*b*.—DESULPHATIZATION OF Ag_2SO_4.

13. *Calcium Sulphate*, $CaSO_4 + 2H_2O$.

The mineral gypsum forms monoclinic white crystals with a pearly to shiny luster, the prepared salt a white voluminous crystalline precipitate. According to Kraut and Precht,[102] $CaSO_4 + 2H_2O$ loses its water at 100°, while Millon [103] maintains that it loses 1.5 molecules of H_2O at 110° and is dehydrated at 300°. Le Chatelier [104] found that gypsum was dehy-

[102] Liebig's *Annalen*, vol. clxxviii., p. 129 (1875).

[103] *Annales de Chimie et de Physique*, Third Series, vol. xix., p. 222 (1847).

[104] *Annales des Mines*, Eighth Series, vol. xi., p. 345 (1887).

drated at 160°. The research of van't Hoff[105] has shown that gypsum heated in a closed vessel to 107° at a pressure of 970 mm. lost 1.5 molecules of H_2O, or rather that it formed the compound $2CaSO_4 + H_2O$. This is confirmed by Le Chatelier.[106] According to Davis,[107] $CaSO_4 + 2H_2O$ undergoes a transformation upon heating before it gives off any water. Above 600° $CaSO_4$ is dead-burnt; that is, it cannot take up any more water (Rohland).[108] Potilizin[109] found that $CaSO_4$ can recombine with water only if it has not been heated to above 200°.

The chemical analysis of the substance used in the experiments is given in Table XIV.

TABLE XIV.—*Composition of* $CaSO_4 + 2H_2O$.

Component.	Chemical Analysis.	Calculated Analysis.
	Per Cent.	Per Cent.
SO_3	46.99	47.07
CaO	32.00	32.01
H_2O	21.01 [a]	20.92
SO_3 : CaO	1.4684	1.4704

[a] By difference.

The lime was precipitated with ammonium oxalate, the calcium oxalate ignited and weighed as CaO; the sulphur was precipitated from a chloride solution with barium chloride and weighed as $BaSO_4$.

In the dehydration experiments there was used 0.2 g. of substance; the charged boat weighed 4.2402 g. The bottom curve, Fig. 24, covers the temperature-range from 16° to 110°. No loss in weight was experienced up to 30°. This was confirmed by holding another sample just below 30° for a considerable time. At 30° is seen the first retardation, which extends to 77°. The second thermal effect is seen at 80°; it extends to 110°. The charged boat heated to 110° weighed

[105] *Zeitschrift für physikalische Chemie*, vol. xlv., p. 257 (1903).

[106] *Comptes rendus de l'Académie des Sciences*, vol. xcvi., No. 23, p. 1668 (1883).

[107] *Journal of the Society of Chemical Industry*, vol. xxvi., No. 13, p. 727 (July 15, 1907).

[108] *Zeitschrift für anorganische Chemie*, vol. xxxvi., No. 3, p. 332 (Sept. 16, 1903).

[109] *Journal der russischen physikalisch-chemischen Gesellschaft*, vol. xxvi., p. 170 (1894).

4.2233 g.; holding for 30 min. at this temperature reduced the weight to 4.2090 g. There had been expelled 0.0312 g. of H_2O, which corresponds to 1.5 molecules of H_2O (= 0.0313 g.). Tests in the thermostat at 33° for 40 hr. showed no loss in water. The retardation at 30° must be ascribed to a transformation followed by an expulsion of water. In the thermostat 1 molecule of H_2O was expelled at 38°; this is indicated in the curve by a very slight retardation. In the upper curve, representing the temperature-range from 110° to 344°, there

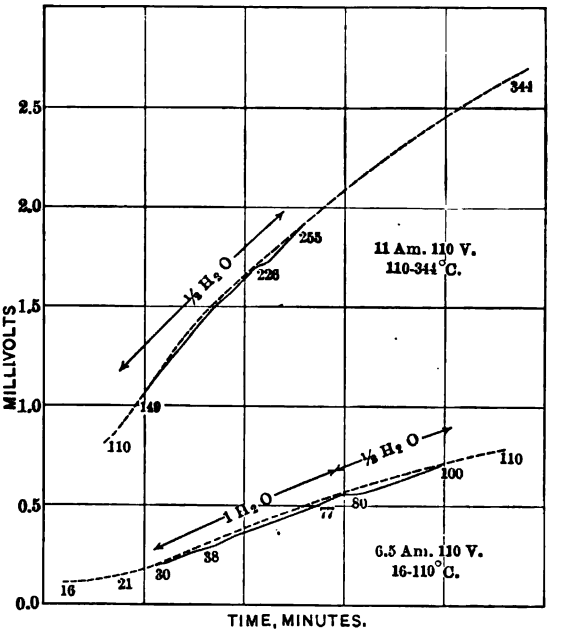


FIG. 24.—DEHYDRATION OF $CaSO_4 + 2H_2O$.

is seen a retardation, which begins at 149° and stops at 255°; it represents the expulsion of the remaining 0.5 molecule of H_2O. The boat heated to 344° weighed 4.2056 g.; heating for 1 additional hour at 340° reduced the weight to 4.2054 g. There had been expelled 0.0348 g., which corresponds to 2 molecules of H_2O (= 0.0348 g.).

The dissociation of anhydrous $CaSO_4$ has been studied by Hofman and Mostowitsch,[110] who found that dissociation began at 1,200°; that the salt fused at 1,360°, being more or less de-

[110] *Trans.*, xxxix., 628 (1908).

composed; that the presence of SiO_2 lowered the beginning of the dissociation to 1,000°, and that the formation of $CaSiO_4$ was finished at 1,250°; that Fe_2O_3 acted similarly to SiO_2, the dissociation beginning at 1,100°, and finishing at 1,250°.

14. *Barium Sulphate*, $BaSO_4$.

The mineral barite forms white orthorhombic crystals having a vitreous to resinous luster; the artificial product is a white crystalline substance when precipitated from a hot acid solution.

The dissociation of this salt has been studied by Mostowitsch,[111] who found that the prepared salt gives off SO_3 at 1,510°, fuses at 1,580°, but is not completely decomposed even then, as the fused mass is a mixture of BaO and $BaSO_4$. He ascertained that the presence of a small amount of foreign substance lowered the dissociation-temperature; also that SiO_2 caused decomposition to begin at 1,000°, and that with Fe_2O_3 it started at 1,100°.

V. Hydrous Sulphates in General.

Our experiments have shown that in subjecting a hydrous sulphate to a gradually increasing temperature all the water of crystallization is split off and carried away by the current of dry air passing through the heating-tube. This separation of water does not take place continuously, but in a series of steps. Thus it has been shown that at a given temperature a sulphate with a certain amount of water of crystallization exists, independently of the other hydrous salts that the sulphate is capable of forming. The hydrous sulphate is a definite chemical compound which takes part as such in chemical reactions. An interesting fact, noted first by Graham[112] and brought out in these experiments, is that, as a rule, the last molecule of H_2O requires for its separation a much higher temperature than do the others, which usually are split off below the boiling-point of water, and which are separated from one another by small intervals of temperature.

The results of our experiments upon dehydration are brought together in Table XV.

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[111] *Metallurgie*, vol. vi., No. 14, p. 450 (July 22, 1909).

[112] *Philosophical Magazine*, Third Series, vol. vi., No. 35, p. 327 (May, 1835).

TABLE XV.—*Dehydration of Metallic Sulphates.*

	Temperature of Beginning of Dehydration.	Product Formed.	Remarks.
	Degrees C.		
$FeSO_4 + 7H_2O$	21	$FeSO_4 + 4H_2O$	Slight apple-green.
$FeSO_4 + 4H_2O$	80	$FeSO_4 + H_2O$	White.
$FeSO_4 + H_2O$	406	$Fe_2O_3 + 2SO_3$	Yellowish brown.
$Al_2(SO_4)_3 + 16H_2O$	51	$Al_2(SO_4)_3 + 13H_2O$	White.
$Al_2(SO_4)_3 + 13H_2O$	82	$Al_2(SO_4)_3 + 10H_2O$	White.
$Al_2(SO_4)_3 + 10H_2O$	97	$Al_2(SO_4)_3 + 7H_2O$	White.
$Al_2(SO_4)_3 + 7H_2O$	109	$Al_2(SO_4)_3 + 4H_2O$	White.
$Al_2(SO_4)_3 + 4H_2O$	180	$Al_2(SO_4)_3 + H_2O$	White.
$Al_2(SO_4)_3 + H_2O$	316	$Al_2(SO_4)_3$	White.
$CuSO_4 + 5H_2O$	27	$CuSO_4 + 3H_2O$	Sky-blue.
$CuSO_4 + 3H_2O$	93	$CuSO_4 + H_2O$	Pale blue.
$CuSO_4 + H_2O$	155	$CuSO_4$	White.
$MnSO_4 + 5H_2O$	25	$MnSO_4 + 2H_2O$	Pale peachblossom.
$MnSO_4 + 2H_2O$	60	$MnSO_4 + H_2O$	Paler than preceding.
$MnSO_4 + H_2O$	152	$MnSO_4$	Paler than preceding.
$ZnSO_4 + 7H_2O$	25	$ZnSO_4 + 6H_2O$	White.
$ZnSO_4 + 6H_2O$	28	$ZnSO_4 + 2H_2O$	White, granular.
$ZnSO_4 + 2H_2O$	115	$ZnSO_4 + H_2O$	White.
$ZnSO_4 + H_2O$	225	$ZnSO_4$	White.
$NiSO_4 + 7H_2O$	40	$NiSO_4 + 4H_2O$	Green.
$NiSO_4 + 4H_2O$	106	$NiSO_4 + H_2O$	Yellow.
$NiSO_4 + H_2O$	279	$NiSO_4$	Orange colored.
$CoSO_4 + 7H_2O$	19	$CoSO_4 + 4H_2O$	Rose.
$CoSO_4 + 4H_2O$	58	$CoSO_4 + H_2O$	Lilac.
$CoSO_4 + H_2O$	276	$CoSO_4$	Lilac.
$CdSO_4 + \frac{8}{3}H_2O$	30	$CdSO_4 + 2H_2O$	White.
$CdSO_4 + 2H_2O$	41	$CdSO_4 + H_2O$	White.
$CdSO_4 + H_2O$	170	$CdSO_4$	White.
$MgSO_4 + 7H_2O$	19	$MgSO_4 + 6H_2O$	White.
$MgSO_4 + 6H_2O$	38	$MgSO_4 + 2H_2O$	White.
$MgSO_4 + 2H_2O$	112	$MgSO_4 + H_2O$	White.
$MgSO_4 + H_2O$	203	$MgSO_4$	White.
$CaSO_4 + 2H_2O$	38	$CaSO_4 + H_2O$	White.
$2CaSO_4 + 2H_2O$	80	$2CaSO_4 + H_2O$	White.
$2CaSO_4 + H_2O$	149	$2CaSO_4$	

In examining the several hydrous sulphates from the point of view of the periodic system (Mendelejeff table, p. 577), it is seen that the sulphates of each group have certain characteristics. Using the notation of Müller-Erzbach[118] to represent the splitting-off of the several molecules of water, there appears the following:

For group VIII., to which belong the metals Fe, Co, Ni, Cu, we have the formula:

$$\frac{MSO_4 + 1H_2O}{+ nH_2O} + nH_2O, \text{ with } n = 3 \text{ for Fe, Co, Ni, and } n = 2 \text{ for Cu.}$$

[118] *Annalen der Physik und Chemie*, vol. xxvi., p. 409 (1885).

Of the 7 molecules of water of $CoSO_4 + 7H_2O$, for example, there are first expelled 3 molecules of H_2O, then 3 more, and finally 1 molecule.

For group VII., to which belongs Mn, we have the formula:

$$\begin{array}{l} MSO_4 + 1H_2O \\ \quad + 1H_2O \end{array} + nH_2O, \text{ with } n = 3$$

Of the 5 molecules of water of $MnSO_4 + 5H_2O$, there are first expelled 3 molecules, then 1 molecule, and again 1 molecule.

For group III., to which belongs Al, we have the formula:

$$\begin{array}{l} MSO_4 + 1H_2O \\ \quad + nH_2O \\ \quad\quad + nH_2O \\ \quad\quad\quad + nH_2O \end{array} + nH_2O + nH_2O, \text{ with } n = 3$$

Of the 16 molecules of water of $Al_2(SO_4)_3 + 16H_2O$, there are expelled 3 molecules in each of the 5 different stages, when the last molecule follows suit.

For group II., to which belong Mg, Ca, Zn, Cd, we have the general formula:

$$\begin{array}{l} MSO_4 + 1H_2O \\ \quad + 1H_2O \\ \quad\quad + 1H_2O \end{array} + nH_2O, \text{ with } n = 4$$

Of the 7 molecules of water of $MgSO_4 + 7H_2O$, there is first expelled 1 molecule, then follow 4, afterward are split off separately the 6th and the 7th.

It is further seen that in each group there is found with the rise of the atomic weight a fall in the water of crystallization. Thus in group VIII., (Fe, Ni, Co)SO_4 have 7 molecules of H_2O, while $CuSO_4$ has only 5; in group II. (Mg, Zn) SO_4 have 7 molecules, while $CdSO_4$ has only $\frac{8}{3}$ molecules. The metals having the largest atomic weights, Ag, Pb, and Ba, are anhydrous.

In each group the even-numbered series has less water of crystallization than the odd-numbered. Thus in group II, (Ca, Ba)SO_4 have less combined water than (Mg, Zn, Cd)SO_4.

VI. Anhydrous Sulphates in General.

The amounts of heat necessary to decompose the several anhydrous sulphates vary greatly; some are dissociated at low

temperatures, others at high. The results of our experiments are assembled in Table XVI., in which they are arranged in the order in which they are dissociated with increasing temperatures. In one column are given the temperatures at which decompositions start, in another the temperatures at which they become energetic; the second column, therefore, gives temperatures which are available for furnace-work. The table shows that some sulphates are directly converted into oxides, while the dissociation of others first leads to a basic salt stable at the temperature at which the first SO_3 is driven off. The basic sulphates obtained in the present series of experiments are: $Fe_2O_3.2SO_3$; $5Bi_2O_3.4(SO_3)_3$; $2CuO.SO_3$; $6PbO.5SO_3$; $3ZnO.2SO_3$; $5CdO.SO_3$.

TABLE XVI.—*Desulphatization of Anhydrous Metallic Sulphates.*

Metallic Sulphates.	Temperature of Beginning of Decomposition.	Temperature of Energetic Decomposition.	Products of Decomposition.	Remarks.
	Degrees C.	Degrees C.		
$FeSO_4$	167	480	$Fe_2O_3.2SO_3$	Yellow-brown.
$Fe_2O_3.2SO_3$	492	560	Fe_2O_3	Red.
$Bi_2(SO_4)_3$	570	639	$5Bi_2O_3.4(SO_3)_3$	White.
$Al_2(SO_4)_3$	590	639	Al_2O_3	White.
$PbSO_4$	637	705	$6PbO.5SO_3$	White.
$CuSO_4$	653	670	$2CuO.SO_3$	Orange-color.
$MnSO_4$	699	790	Mn_3O_4	Dark red to black.
$ZnSO_4$	702	720	$3ZnO.2SO$	White, cold and not.
$2CuO.SO_3$	702	736	CuO	Black.
$NiSO_4$	702	764	NiO	Brownish green.
$CoSO_4$	720	770	CoO	Brown to black.
$3ZnO.2SO_3$	755	767	ZnO	Hot yellow, cold white.
$CdSO_4$	827	846	$5CdO.SO_3$	White.
$5Bi_2O_3.4(SO_3)_3$	870	890	Bi_2O_3(?)	Yellow.
$5CdO.SO_3$	878	890	CdO	Black.
$MgSO_4$	890	972	MgO	White.
Ag_2SO_4	917	925	Ag	Silver white.
$6PbO.5SO_3$	952	962	$2PbOSO_3$(?)	White to yellow.
$CaSO_4$	1,200	.....	CaO	White.
$BaSO_4$	1,510	.....	BaO	White.

Considering the formation of basic salt from the point of view of the periodic system, it appears that in each group the tendency to form basic salts increases with the rise of the atomic weight. Thus in group VIII., in the iron series, Cu, with the largest atomic weight, forms a basic salt, while Fe, with the smallest atomic weight, forms no ferrous basic salt; normal ferric salt also forms no basic salt. In groups V. and

IV., the metals with the highest atomic weights, that is, Bi and Pb, form basic salts. In group II., $MgSO_4$ is dissociated into MgO and SO_3, while (Zn, Cd)SO_4 form basic salts.

Independently of the periodic system, there will be noted further characteristics: Normal sulphates which do not form basic salts upon dissociation show both SO_2 and SO_3 in the gaseous product; normal sulphates which form basic salts upon dissociation evolve only SO_2 when they pass from the normal to the basic state; basic sulphates upon dissociation into MO set free only SO_3; metallic sulphates with bases M_2O_3 and M_2O do not form basic sulphates upon dissociation; the former set free SO_3, the latter SO_2.

Mendelejeff Table.

Group.	I.	II.	III.	IV.	V.	VI.	VII.	VIII.
Series.								Composition of Hydrogen Compounds.
1.	H 1			RH^4	RH^5	RH^3	RH	
2.	Li 7	Be 9.1	B 11	C 12	N 14	O 16	F 19	
3.	Na 23	Mg 24.4	Al 27	Si 28	P 31	S 32	Cl 35.5	
4.	K 39.1	Ca 40	Sc 44	Ti 48.1	V 51.2	Cr 52.3	Mn 55	Fe 56, Co 59.1, Ni 58.5, Cu 63.3
5.	(Cu) 63.3	Zn 65.4	Ga 69.9	Ge 72.5	As 75	Se 79	Br 80	
6.	Rb 85.4	Sr 87.5	Y 89	Zr 90.7	Nb 93.5	Mo 95.9		Ru 101.7, Rh 102.9, Pd 106.7, Ag 107.9
7.	(Ag) 107.9	Cd 112.4	In 114.8	Sn 119	Sb 120.2	Te 125.2	I 126.9	
8.	Cs 132.8	Ba 137	La 138.5	Ce 141.5	Di ? 145			
9.								
10.			Yb 173.2		Ta 182.8	W 184		Os 190.9, Ir 193.1, Pt 195.2, Au 197.2
11.	(Au) 197.2	Hg 200	Tl 204	Pb 207.1	Bi 208			
12.				Th 232.4		U 238.5		
	R^2O	R^2O^2	R^2O^3	R^2O^4 RO^2	R^2O^5	R^2O^6 RO^3	R^2O^7	Composition of Highest Saline Oxides. RO^4

The Direct Determination of Small Amounts of Platinum in Ores and Bullion.*

BY FREDERIC P. DEWEY, WASHINGTON, D. C.

(New York Meeting, February, 1912.)

By the old method of determining platinum in ores and bullion, the silver-alloy first obtained in the regular course of assay is parted in strong sulphuric acid and the residual metal weighed. This is re-alloyed with silver by a second cupellation and parted in nitric acid, the residual metal being again weighed. Any difference shown between the two weighings is assumed to be, and is called, platinum. Sometimes it is so, and if any considerable amount of platinum be present, there will be a decided difference between the two weighings; but a slight difference is no real evidence whatever of the presence of platinum. On the other hand, the second weight may equal or possibly exceed the first, even when traces of platinum are present. Again, other members of the platinum group may go into solution in nitric acid more or less. If present, these would be called platinum and escape detection. The method does not provide any direct tests whatever as to the presence or absence of platinum. It is often indecisive, and sometimes gives erroneous results. It is, therefore, quite unsatisfactory.

Being called upon many times to determine platinum in a wide variety of materials, particularly when present in very small amounts, I have realized the disadvantages and defects of this old method.

In an article on the solubility of gold in nitric acid,[1] I have briefly outlined a method of gathering a little gold out of a solution containing much silver, which furnishes the basis of

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* Published by permission of the Director of the Mint. Published also by permission of the Council in *The Journal of Industrial and Engineering Chemistry.*

[1] *Journal of the American Chemical Society*, vol. xxxii., No. 3, p. 318 (Mar., 1910).

an excellent method for the direct and absolute determination of small amounts of platinum, which has the added advantage that the metal weighed may be subjected to suitable tests, to determine that it really is platinum, and to reveal the presence of other members of the platinum group.

In the regular course of assaying for the precious metals, gold is parted from silver by dissolving the silver in nitric acid. If platinum be present in small amounts only, it will readily go into solution in the nitric acid. If now a limited amount of hydrogen sulphide be added to the solution from parting, any platinum present will be precipitated as sulphide, along with some silver sulphide. On filtering off the precipitate (which generally is sufficiently washed by the operations necessary to transfer it from the precipitating-dish to the filter), the moist filter is transferred to a small porcelain crucible, dried at a low heat, and burned off by gentle ignition. This transforms the sulphide precipitate into a metallic sponge, which is wrapped in a small piece of thin lead foil and cupelled. The resulting bead is then parted in strong sulphuric acid, when the platinum will be left as a dark residue, generally collected in spongy form, even when minute in quantity. This sponge, after reboiling in fresh acid, if necessary, is suitably washed by decantation, annealed, and weighed.

Generally, the final metal speaks for itself as being platinum, but, if there should be any doubt, it may be dissolved in a drop or two of aqua regia and gently evaporated. The solution obtained may be tested with potassium iodide, or a few small crystals of ammonium chloride may be added, when the characteristic precipitate will show itself. As a further test, this may be filtered off and gently ignited to produce spongy platinum. If the amount of the final metal be considerable, the platinum may be determined by the double-chloride method. Any decided difference shown would indicate the presence of other members of the platinum group, for which direct test could then be made.

For precipitating the platinum and the necessary silver from the parting-solution, a very dilute solution of hydrogen sulphide should be used. One part of a strong solution should be diluted to from 10 to 20 parts with water. If the solution of silver nitrate be strongly acid, it should be largely diluted, or

it may first be evaporated and then diluted. The very dilute hydrogen sulphide solution should be added very slowly to the silver nitrate solution with constant stirring. The solution is, of course, at once darkened, but there should be no immediate separation of a visible precipitate. The solution should be stirred occasionally, and in about 2 hr. flocks of precipitate should appear. It may be filtered in from 3 to 4 hr., but it is a good plan to let it stand over night if possible.

The amount of hydrogen sulphide required depends, of course, upon the amount of platinum present. If this should be roughly known or suspected, the amount used should generally be enough to precipitate the platinum and from three to five times as much silver. On an entirely unknown ore, I should at first use 1 cc. of strong hydrogen sulphide solution diluted to 15 cc., and reserve the filtrate from the sulphides for retreatment, if necessary. On an unknown bullion I should use 2 cc. of strong solution diluted to 30 cc., partly because bullions are liable to carry much more platinum than any ordinary ore, and partly because the volume of the silver nitrate solution from parting the gold must necessarily be larger. If, however, it is known that minute amounts of platinum are present, it is still necessary to use sufficient hydrogen sulphide to give a silver bead large enough to handle comfortably. For this reason I seldom use less than the equivalent of 1 cc. of strong hydrogen sulphide solution.

It may happen that the final metal shows the yellow color of gold, due to the fact that exceedingly fine float-gold passed over in decanting the solution of silver nitrate from the gold. In such a case the metal must be re-alloyed with silver and the treatment repeated. When the proportion of gold to silver in the metal being parted is so small that the gold separates in a very finely divided state, it will often save trouble to filter the silver nitrate solution, to separate any float-gold, before adding the hydrogen sulphide.

This method has been used with the utmost satisfaction in determining very minute amounts of platinum in various silver-products directly. Much of our silver coinage, for instance, will show a few tenths of a milligram of platinum in 100 g. of coin. Recently I examined samples from two purchases of fine silver. Very large samples were dissolved in nitric acid.

The acid in portions was poured upon the samples and allowed to act at a gentle heat until exhausted. Finally, a small amount of residual silver was removed from the solution and dissolved in a small amount of fresh acid, the solution being then united with the main solution, and the whole evaporated nearly to dryness. It was then diluted to about 250 cc., and 5 cc. of strong hydrogen sulphide solution diluted to 50 cc. was poured in with constant stirring.

This operation concentrated the gold and platinum of the silver into a small amount of sulphide precipitate. This precipitate was filtered off, roasted, and cupelled. The resulting bead was parted in nitric acid, and the gold was determined. The silver nitrate solution was treated with dilute hydrogen sulphide solution, equivalent to about 1 cc. of strong solution, and the platinum parted from the silver by strong sulphuric acid.

These two samples yielded the following results:

	Silver Taken. Grams.	Gold Found. Milligram.	Platinum Found. Milligram.
No. 1, . .	122.32	0.28	0.67
No. 2, . .	125.47	0.12	0.18

In case we have a material containing a considerable amount of platinum, the well-known fact that platinum alloyed with silver is not entirely soluble in nitric acid must be considered. In such a case the gold from the first parting in nitric acid must be alloyed with silver and parted in nitric acid a second, or even a third, time, before proceeding to precipitate the platinum from the parting-solutions with hydrogen sulphide.

It is also very satisfactory to use the general method of gathering gold in a precipitate of silver sulphide in determining minute quantities of gold in high-grade silver, such as that produced by electrolytic refining. It is comparatively easy to gather the gold from very large samples of silver, up to 100 g. or more, into a decigram of silver, and then part by nitric acid as usual.

Probably this method of precipitating a noble metal in solution, or removing it from suspension in a liquid, by adding hydrogen sulphide in the presence of silver in the solution, could be used to advantage in determining gold in metallic copper and similar materials.

The Solubility in Nitric Acid of Gold Contained in Certain Copper-Alloys (Copper-Bullions).

BY EDWARD KELLER, PERTH AMBOY, N. J.

(New York Meeting, February, 1912.)

In a paper, entitled A Uniform Method for the Assay of Copper Material for Gold and Silver,[1] A. R. Ledoux invited the assayers of this country to contribute to a symposium, in which the results of their assays of given samples of copper-matte and metallic copper would be made comparable. This symposium[2] appeared about one year thereafter; and through it and its discussion, first became generally known the fact that the gold-yield by the all-fire method of assaying is usually higher than by the so-called combination (wet and dry) method, in which nitric acid is employed as a solvent for the copper. In the publication alluded to, the supposed cause of gold-losses in the latter method was declared to be the solvent action of copper nitrate, or of the reduction-products of the nitric acid, upon the gold. Up to a very recent date, gold has been held to be insoluble in pure nitric acid. (The solvent action of any chlorine present in the nitric acid must here properly be disregarded as belonging in the realm of carelessness.)

Among later allusions to this topic, W. R. Van Liew[3] ascribes the solution of the gold in the combination-assay entirely to the action of nitrous acid at the high reaction-temperature of copper and nitric acid; F. B. Flinn[4] believes the low result in gold to be due exclusively to mechanical losses—finely-divided gold penetrating through the filter-paper—but presents no experimental data confirmative of this belief; and O. Pufahl[5] attributes the solubility of the gold in some coppers to the presence of selenium—a view which appears to be shared by many as-

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[1] *Trans.*, xxiv., 575 (1894).

[2] *Trans.*, xxv., 250 (1895).

[3] *Engineering and Mining Journal*, vol. lxix., No. 16, p. 469 (Apr. 21, 1900).

[4] *Engineering and Mining Journal*, vol. lxxxvii., No. 11, p. 569 (Mar. 13, 1909).

[5] Lunge, *Chemische Technische Untersuchungs Methoden*, vol. ii., p. 229 (1900).

sayers. There seems, however, to be no foundation in fact for such an assumption. We do know that gold is dissolved by concentrated selenic acid when heated. When copper with minute selenium-content is dissolved in nitric acid, selenious acid is formed, which is highly diluted, and has, to our knowledge, no action whatever on the gold. Recently, F. P. Dewey[6] has convincingly demonstrated that pure gold dissolves in small quantity, on long-continued boiling, in pure concentrated nitric acid.

Before discussing experiments and theories on this subject, it may be well to state the chemical composition of the various auriferous copper-bullions, and briefly to describe their treatment in commercial sampling and assaying, since in this field lies the practical importance of a thorough knowledge of the problems involved.

In shape, copper-bullion castings are generally known as pigs (ingots), and slabs (plates), the former being thick and the latter thin, as compared to their other dimensions. The first form favors and the second minimizes so-called segregation. The weight of the castings ranges approximately between 200 and 600 lb. They are sampled by drilling through their entire thickness—the holes being placed according to regular templates; and the drillings are brought to the proper fineness by grinding. In several of our large copper-refineries the grinding process is carried to a point where the whole sample will pass a 16- or even a 20-mesh screen. In my own practice these samples are separated into coarse and fine parts by means of a 40-mesh screen. It is self-evident that the fine part will contain all such extraneous matter as the oxides, slags, mold-wash and incidental dirt, while the coarse part consists of clean metallic (alloy) particles. Table I. shows differences in metal-values of the fine and coarse parts which demonstrate the necessity of separate assays of the two and the averaging of the results according to the ratio of the parts; or else of making combined assays of the fine and coarse, weighed and mixed in their proper ratio. Some assayers believe they can accomplish this object equally well by reducing the mixed sample to approximately the desired weight by some mechanical cutting-down device.

[6] *Journal of the American Chemical Society*, vol. xxxii., No. 3, p. 318 (Mar., 1910).

As regards the methods of assaying the copper-bullions for gold, little need be said of the all-fire method. It assumes that all the copper is eliminated by scorification and cupellation. In these two operations, gold-losses occur by slagging and absorption; but, if these be determined and accounted for, operations by this method under varied conditions yield concordant results, which are the most accurate attainable. The nitric acid combination-method presents a very different case. Here, variation in heat-conditions, or in the dilution of the acid used in dissolving the copper, causes varying quantities of the gold to be dissolved, and therefore lost.

In the tests embodied in this paper, a uniform procedure was employed. To 1 assay-ton of the copper was added 190 cc. of distilled water with 90 cc. of pure nitric acid (sp. gr. 1.42); and after the violence of the reaction had subsided, the beaker was placed on a steam-plate, at 160° F., where it was left until the red fumes had been expelled. No chloride was used to precipitate the silver; but the gold was gathered on a triple S. & S., No. 597, 12.5-cm. filter covered with 2 g. of test-lead; and, after incineration of the filter and the addition of 20 g. of test-lead, a sufficient quantity of silver, in the form of nitrate crystals, was added. That part of the combination-method designated as "dry" needs no description, since what has been said of the all-fire method holds true here, namely, that when the so-called fire-losses are determined and accounted for, variations in this part of the method do not alter the results.

When, therefore, we have determined the amount of gold by the all-fire method and the amount by the combination-method, the difference should be equal to the gold-loss by solution in the latter method. (The suggestion that this difference may involve other factors will be discussed hereafter.) All the results for gold given in this paper are averages, generally of five assays, often of more. The accuracy of the all-fire method is, almost without exception, within 0.03 oz. per ton, while the combination-method sometimes shows on the higher-grade bullions differences as high as 0.2 oz. per ton. The measurement by the ounce per ton has been employed throughout for silver and gold, not because this is the commercial custom, but because it is much more convenient for comparison than the extended decimal fractions expressing percentages. In carry-

ing out the various assays and analyses, I was assisted by K. W. McComas and Albert Ferrell.

From Table I. it will be seen that, on account of the admixture of extraneous impurities, already referred to, the copper-content in the fines is always lower than in the coarse, while in many cases (class A) the fines show an enrichment in other elements.

The examples of copper-bullions in Table I. have been divided into two classes, A and B. In general, it may be remarked that to class A belong all those of a copper-tenor above 97 per cent., and to class B those of a copper-tenor below that figure. In the castings of class A, the precious metals and other elements are found concentrated towards the freezing-center of the castings, while in class B they are found richer towards the surfaces.[7] None of the elements in question, however, show the same degree of heterogeneity, so that, in some instances, the two classes overlap each other. Viewed as solutions, class A must be considered as copper subsaturated above its freezing-point (hypo-eutectic); while in class B the copper is saturated above its freezing-point (hyper-eutectic). In class A the copper is the element freezing first; in class B the other elements freeze first.

Looking again at the composition of the samples, fine and coarse, it will be noted that under class A the fines, generally, are higher in foreign elements than the coarse; and it is consistent to conclude that the fines (exclusive of extraneous matter), in their greater portion, are derived from the later-freezing parts of the bullion castings. The gold in the fines shows a markedly greater solubility in nitric acid than the gold in the coarse, which, as will be seen later, is in conformity with the supposition that the fines are derived from the last and most slowly-cooled portions of the castings. The fines in class B show a smaller amount of impurities than the coarse, thereby also proving their origin from the last-freezing portions of the bullion. In them, moreover, the gold-solubility is much greater than in the coarse.

The samples of bullions, as regards their metallurgical origin, are as follows: Nos. 1 and 2, converted copper, furnace-refined; Nos. 3, 4, 5, 6, and 7, converted copper, gas-finished;

[7] E. Keller, *Trans.*, xxvii., 106 (1897).

No. 8, reverberatory blister-copper; Nos. 9 and 10, blast-furnace black copper.

My first serious attention to the solubility of gold in nitric acid was drawn by the peculiar facts connected with bullions Nos. 1 and 2, Table I. They were derived from the same source and treated identically in the process of refining; often coming from the same furnaces, which statement finds corroboration in the results of the partial analyses. Both were quenched (pickled) as soon as solidified. Two parallel series of gold-assays were made for many millions of pounds of material; one series being by the all-fire method, the other by the combination-method. The results of these two series of assays were, that for bullion No. 1, the gold-yield by the combination-method showed a deficit of 13.71 per cent., as compared with the all-fire method, while for No. 2 the corresponding figure was 26.62 per cent.—both, it may be repeated, under identical conditions of assaying. Since, as already remarked, all the assays were corrected for the so-called fire-losses, it followed that the results were to be so interpreted that the figures 13.71 per cent. and 26.62 per cent. denoted the solubility-losses of the gold in the two respective bullions—in one nearly twice as great as in the other. Although these two bullions were the same in origin, composition, and metallurgical treatment, there remained one evident difference in the castings, as to size and weight; No. 1 weighing about 300 lb., and No. 2 over 500 lb. This fact implied a difference in the rate of cooling, and led at once to the conclusion that gold contained in slowly-cooled copper is more soluble in nitric acid than that in rapidly-chilled copper. That the gold-solubility was not influenced by any mechanical differences in the two samples is evidenced by the fact that the ratios of fine and coarse in both are very nearly alike, while the size of the coarse particles in both samples is the same, yet the solubility of the gold in the coarse is for No. 1 only one-half of that in No. 2.

If, then, the conclusion reached be true, that the solubility in nitric acid of the gold in the same bullion or alloy is subject to great variation according to the rate of cooling, it should be possible to reduce this solubility to a minimum by the instantaneous chilling of the molten bullion. Granulation, or shotting, by running a stream of the molten bullion into cold

water, suggested itself, and advantage was taken of a blast-furnace charge of copper, for an experiment, to compare the behavior of the gold in a shot-sample with that in a drill-sample from a large and slowly-cooled slab. The results of the tests are incorporated in Table I., bullion No. 10. They demonstrated at once the correctness of the hypothesis; none of the gold in the granules or shot being soluble, while the absolute amount of gold from the slowly-cooled slab dissolved by nitric acid was greater than in any other of the samples of copper-bullion or alloys that have been tested.

Thus far, the conclusions reached were based on the analytical methods; it now appeared desirable to proceed to the synthetical.

Table II. shows the solubility of the gold in shot and drillings of a series of alloys, each of which was prepared by adding 0.5 per cent. of a foreign element and an amount of gold equaling approximately 2 oz. per ton to commercial electrolytic copper. The alloy was thoroughly fused and mixed under a charcoal cover in a crucible. About 3,000 g. of the alloy was cast into a heated mold to form a small ingot; the remainder being granulated. Each ingot was drilled in the same manner for a sample, while the granules could directly be used for analysis. It would be venturing too much to claim for the solubility-figures in Table II. that they are the true relative expressions of the influence of each of the given elements on the solubility of the gold contained in the ingots, for there was, probably, a wide variation between them in the conditions of cooling and freezing. In the process of granulating it is far more likely that those conditions are nearly alike in all cases, and that here the effects of the alloyed elements on the gold-solubility are shown in their proper relative degrees. It seems to be demonstrated that the quenching effect diminishes from the metallic towards the metalloidal end of the series, reaching zero with selenium.

The largest quantity of soluble gold thus far found is 0.495 oz. per ton, or 15.55 per cent. of the total gold in this special case, or 0.0017 per cent. of the total alloy—which is a very small proportion. It became of interest, therefore, to determine if this figure could be materially enlarged by the increase of either the gold- or the impurity-content of the copper-alloy.

Table III. shows the negative results on alloys with varying tenors of gold and arsenic. In these cases small ingots were cast in heated molds, which might not preclude too rapid cooling; therefore, tests of the slowest possible cooling were instituted, one of which is exemplified in Table IV., with negative results. This shows also how the great gold-solubility in the fine portion of a sample is reduced by remelting and granulation.

Since neither gold-increase nor impurity-increase appeared to augment the quantity of soluble gold, it became a question what influence pure copper would have in the solubility of a relatively small gold-content. Table V. gives the results of a test with the purest commercial electrolytic copper to which gold was added by melting. It shows that the gold is as soluble in this as in many of the ordinary bullions; but, since the purest commercial electrolytic copper[8] still contains as much, or more, of arsenic and antimony, per cent., as there is soluble gold in most of the bullions, and absolutely pure copper is not available in quantity, the effect of the latter remains unknown.

The tests on the solubility of gold in nitric acid were also extended to a number of lead-bullions; all with results that closely approached the negative. Table VI. gives those obtained from a comparatively large and slowly-cooled casting.

Reference has already been made in the beginning to the bearing of this subject upon the practical sampling and assaying of auriferous copper-bullions. Ignorance in this field has, in the past, led to some acrimonious controversies, the possibilities of which may be understood most readily through an illustration: A furnace-charge of molten copper would be sampled by shotting (properly done)[9]; then several samples by drilling would be taken from slabs, varying from a thin sample plate to the largest-sized commercial slab. The gold-assays on all the samples would now, by one party, be made by the all-fire method, with concordant results, and with the verdict that all the methods of sampling are correct. A second party, believing the combination-method reliable, would, by its use, find concordant results between the shot-sample and the thin-plate

[8] E. Keller, *Mineral Industry*, vol. viii. (1899).

[9] On the pitfalls of sampling by shotting see: E. Keller, *Trans.*, xxvii., 106 (1897). Wm. Wraith, *Trans.*, xli., 318 (1910).

sample, while with the other samples the results would show an increasing deficit, reaching its maximum, perhaps 30 per cent., with the sample from the largest slab. This party's pronouncement would be: shot and thin-plate sampling correct; samples from large casting absolutely wrong.

It may be observed in passing that the accuracy in the gold-assay has been multiplied many times during the past decade through the remarkable improvement in the construction of gold-balances.

At the present day there probably exist but few contracts in the copper-world which permit the employment of the nitric acid combination-method for the assay of gold in copper-bullions, although this method may still be pronounced quite permissible for quick routine-work, when properly modified, and, especially, when applied to quenched or otherwise rapidly-cooled samples.

Summary.

I. *Observed Facts.*—Gold contained in copper-alloys, or bullions, is soluble in nitric acid in varying degrees; the solubility diminishing with the increased rate of cooling of the castings, and reaching a minimum when the molten bullion is quenched in cold water. In nearly all of the quenched samples the gold, determined by the combination-method of assaying, still shows a deficit as compared with the amount found by the all-fire assay; the maximum of this deficit being approximately 2 per cent.

The highest amount of soluble gold was found to be 0.0017 per cent. of the total bullion; no increase in gold- or impurity-content was able to augment that amount.

A definite relation between gold-solubility and impurity-content of the bullions has not been established; the former being more dependent on the rate of cooling than on the quality or quantity of impurity. This proposition is demonstrated by the fact that the gold-solubility in all samples is greatest in the fine portions, which in classes A and B are the most slowly-cooled parts (this conclusion being deduced from the chemical composition). In class A the fine portion is more impure than the coarse; in class B this relation is reversed.

The observations by Dewey of the solubility of pure gold in boiling concentrated nitric acid, and those of Van Liew for

avoiding the solution of gold in copper-bullion by dissolving the latter in very dilute nitric acid under ice-cooling, are not controverted.

II. *Hypothetical Conclusions.*—The smallness of the absolute amount of soluble gold contained in bullions makes it doubtful whether the actual condition of this gold can, at present, be determined by any of the known methods, chemical or physical.

From what is known of the action of strong nitric acid, or other chemicals, on pure or uncombined gold, it would follow that these would not discriminate between such gold obtained in slowly-cooled and in quenched bullions. It is, furthermore, not probable that nitric acid, of the dilution used in dissolving these bullions, and in the presence of a great excess of metallic copper, would attack uncombined gold. The difference in the properties of the gold in slowly-cooled and in quenched bullions will, therefore, rationally be sought in a change of chemical conditions, or of the chemical nature of the alloy itself, by the heat-treatment. It seems justifiable to assume that a portion of the gold in slowly-cooled bullion combines with some, or all, of the accompanying elements, forming compounds soluble in nitric acid; that the reaction takes place through a narrow range of temperature above the freezing-point, and that it is a slow one as compared to the speed of cooling; that at higher temperature the gold-compounds dissociate; and that quenching of the bullions at those temperatures leaves the gold in the uncombined state, and, therefore, insoluble in the nitric acid used to dissolve the bullion. Since neither gold- nor impurity-increase augments the amount of soluble gold beyond a low maximum, it would appear that the soluble gold-compounds are formed only in very dilute solutions, or under low osmotic pressure.

The fact that with the seleniferous alloy quenching has no reducing effect on the gold-solubility, may be explained by the supposition that probably the gold-selenium compound forms at higher temperatures than the others, or that it has not dissociated at the temperatures obtaining in the furnace or crucible at which quenching is performed.

The 2 per cent. or less of gold-deficit, as found by the combination-assay in quenched bullions, may be due to a re-

sidual soluble gold-compound, formed on account of the impossibility of instantaneous chilling by quenching. Should this explanation be unacceptable, then it could be conceded that this loss is due to all other causes that have been suggested for the loss of uncombined gold.

Phenomena of similar nature to those suggested in the foregoing have been well established by scientific investigation. C. T. Heycock and F. H. Neville [10] have demonstrated that gold in metallic solution forms chemical compounds with metals, as for instance, tin as a solvent, gold and cadmium combine in the alloy to form in succession the following compounds: Au_4Cd_4, Au_5Cd_5, Au_9Cd_9. One of the instances best known to metallurgists of a chemical compound being formed at a certain temperature and being dissociated at higher ones, is the case of iron carbide (cementite) in steel. The carbide is formed when steel is slowly cooled, and its formation is prevented when the steel is quenched above the dissociation-temperature; chemical conditions are here changed by heat-treatment. In other respects there is much dissimilarity between the carbide in steel and the hypothetical gold-compounds in copper-bullion. The carbide is formed well below the freezing-point of the steel; it is not attacked by dilute sulphuric and hydrochloric acids, while in quenched steel the dissociation-products readily combine; the iron to form a salt, the carbon a hydride. In the quenching process the heat of combination and crystallization remained latent and this becomes instrumental in forming the compound with the acids. An equally well-known case in which, by heat-treatment, a physical change only is wrought, is that of the quenched blast-furnace slags. When these are permitted to cool slowly they attain the crystalline state and become insoluble in acids, while when the molten slags are run into cold water they remain amorphous; the heat of crystallization remains latent, and they are readily attacked by the acids.

[10] *Journal of the Chemical Society* (London), vol. lix., p. 936 (1891).

TABLE I.—*Copper-Bullions: Character; Weight of Castings; Partial Analyses; Gold-Solubility in Nitric Acid.*

Class.	Remarks.	Samples.	Ratio. Fine / Coarse.	Cu.	Sb.	As.	Ag.	Gold. Assay-Method. Fire.	Gold. Assay-Method. Comb.	Gold. Difference.	Gold-Solubility.
				Per Cent.	Per Cent.	Per Cent.	Oz. Per Ton.	Oz. Per Ton.	Oz. Per Ton.	Oz. Per Ton.	Per Cent.
A.	No. 1. Converter; Refined; 300 lb.	Fine.....	1 / 20.07	98.642	0.0349	0.0392	81.01	0.415	0.218	0.197	47.47
		Coarse.....		99.307	0.0254	0.0395	81.22	0.421	0.370	0.051	12.11
		Average..		99.276	0.0259	0.0395	81.21	0.4207	0.363	0.0577	13.71
	No. 2. Converter; Refined; 500 lb.	Fine	1 / 17.11	98.654	0.0359	0.0489	80.45	0.415	0.202	0.213	51.32
		Coarse.....		99.346	0.0261	0.0474	80.13	0.401	0.300	0.101	25.19
		Average..		99.308	0.0266	0.0475	80.14	0.402	0.295	0 107	26.62
	No. 3. Converter; Gas finish; 225 lb.	Fine..	1 / 14.26	98.130	0.0531	0.0542	45.30	0.292	0.124	0.168	57.53
		Coarse		99.342	0.0483	0.0434	41.90	0.278	0.246	0.032	11.51
		Average..		99.263	0.0486	0.0441	42.14	0.279	0.238	0.041	14.70
	No. 4. Converter; Gas finish; 230 lb.	Fine	1 / 28.52	98.139	0.0610	0.0743	95.13	0.618	0.379	0.239	38.67
		Coarse.....		98.860	0.0448	0.0219	87.52	0.678	0.633	0.045	6.64
		Average..		98.627	0.0455	0.0240	87.83	0.676	0.623	0.053	7.84
	No. 5. Converter; Gas finish; 205 lb.	Fine.....	1 / 8.50	95.756	0.0289	0.0338	64.67	0.0707	0 0240	0.0467	66.05
		Coarse,....		98.896	0.0305	0.0220	63.26	0.0707	0.0420	0.0287	40.59
		Average..		98.566	0.0303	0.0232	63.41	0.0707	0.0401	0.0306	43.28
	No. 6. Converter; Gas finish; 230 lb.	Fine.....	1 / 10.25	96.046	0.0079	0.0077	69.32	2.264	1.578	0.686	30.30
		Coarse.....		98.858	0.0111	0.0036	66.89	2.274	2.178	0.096	4.22
		Average ..		98.608	0.0108	0.0040	67.11	2.273	2.125	0.148	6.51
	No. 7. Converter; Gas finish; 400 lb.	Fine.....	1 / 3.149	98.350	.....	0.2120	88.71	22.821	22.7798	Within the limits of error.	
		Coarse.....		98.742	.....	0.2069	83 39	22.1208	22.1615		
		Average..		98.648	.....	0.2081	84.67	22.2897	22.3105		
	No. 8. Reverberat. Blister	Fine.....	1 / 11.50	96.482	0.0186	0.0071	60.73	3.992	3.315	0.677	16.96
		Coarse.....		99.120	0.0138	0.0076	55.32	4.275	4.047	0.228	5.33
		Average..		98.905	0.0142	0.0070	55.75	4.252	3.988	0.264	6.21
B.	No. 9. Blast-furnace; Black....	Fine.....	1 / 9.89	95.107	0.0379	0.0776	11.04	0.403	0.174	0.229	56.82
		Coarse.....		97.407	0.0568	0.1225	11.07	0.385	0.364	0.021	5.45
		Average..		97.196	0.0551	0.1184	11.067	0.387	0.347	0.040	10.34
	No. 10. Blast-furnace; single slab 1.5 in. from edge	Fine.....	1 / 2.84	89.757	1.358	0.981	176.10	3.105	2.327	0.778	25.06
		Coarse.....		89.980	0.932	1.005	177.80	3.170	2.797	0.373	11.77
		Average..		89.922	1.043	0.999	177.36	3.153	2.675	0.478	15.16
	Same; 3 in. from edge.....	Fine.....	1 / 3.81	89.572	1.279	0.958	174.70	3.120	2.579	0.541	17.34
		Coarse, ..		90.096	0.829	0.971	175.90	3.180	2.999	0.181	5.69
		Average..		89.989	0.937	0.968	175.44	3.167	2.912	0.255	8.05
	Same; 4.5 in. from edge.....	Fine.....	1 / 6.45	89.266	1.074	0.929	176.10	3.110	2.239	0.871	28.01
		Coarse.....		90.108	0.742	1.048	175.70	3.195	2.759	0.436	13.65
		Average..		89.984	0.787	1.032	175.75	3.184	2.689	0.495	15.55
	Same; shot, or granules.....	.....	.....	.....	.....	.....	.....	3.000	3.039	Within the limits of error.	

NOTE.—The amount of copper taken in the analysis for antimony and arsenic was from 10 to 25 g., depending on the grade of the metal.

TABLE II.—*Alloys of Copper, Granulated, and Cast into Bars; Showing Various Nitric Acid Solubilities of Gold.*

Element Alloyed. 0.5 Per Cent.	Granules (Shot). Gold. Assay-Method. Fire.	Granules (Shot). Gold. Assay-Method. Comb.	Granules (Shot). Difference.	Granules (Shot). Gold-Solubility.	Bar-Drillings. Gold. Assay-Method. Fire.	Bar-Drillings. Gold. Assay-Method. Comb.	Bar-Drillings. Difference.	Bar-Drillings. Gold-Solubility.
	Ounces Per Ton.	Ounces Per Ton.	Ounce Per Ton.	Per Cent.	Ounces Per Ton.	Ounces Per Ton.	Ounce Per Ton.	Per Cent.
Pb	1.934	1.8985	0.0355	1.84	1.954	1.6965	0.2575	13.18
Bi	1.927	1.890	0.037	1.92	1.933	1.789	0.144	7.45
Sb	1.900	1.832	0.068	3.58	1.890	1.676	0.214	11.32
As	1.894	1.826	0.068	3.06	1.874	1.592	0.282	15.05
Te	1.923	1.848	0.075	3.90	1.910	1.748	0.162	8.48
Se	1.935	1.856	0.079	4.08	1.935	1.860	0.075	3.88

TABLE III.—*Solubility in Nitric Acid of Gold in Miscellaneous Copper-Arsenic Alloys.*

Arsenic.	Drillings From 3,000-g. Bars. Assay-Method; Gold. Fire.	Comb.	Difference.	Gold-Solubility.
Per Cent.	Ounces Per Ton.	Ounces Per Ton.	Ounce Per Ton.	Per Cent.
1.680	21.381	21.245	0.136	0.636
1.110	2.018	1.965	0.053	2.626
0.767	11.081	10.827	0.254	0.290
0.511	7.9497	7.874	0.0957	1.204
0.369	6.214	6.140	0.074	1.191
0.225	3.112	3.014	0.098	3.149
0.117	1.576	1.524	0.052	3.300

TABLE IV.—*Copper (62 lb.) Melted in Crucible, Gold and Arsenic Added, and Left to Cool Therein in the Furnace.*

Position of Copper in Crucible.	Sample.	Ratio.	Arsenic.	Gold. Assay-Method. Fire.	Gold. Assay-Method. Comb.	Gold. Difference.	Gold-Solubility
			Per Cent.	Ounces Per Ton.	Ounces Per Ton.	Ounces Per Ton.	Per Cent.
Top third...	Fine.....	1	1.423	7.521	6.270	1.251	16.633
	Coarse	6.037	1.165	7.226	7.116	0.110	1.522
	Average.....		1.201	7.2679	6.9957	0.2722	3.745
Center third	Fine.....	1	1.013	6.522	5.955	0.567	8.694
	Coarse.....	6.835	0.864	6.432	6.370	0.062	0.964
	Average. ...		0.883	6.4434	6.3170	0.1264	1.962
Bottom third	Fine.....	1	0.959	6.422	6.040	0.382	5.948
	Coarse.....	5.574	0.896	6.262	6.193	0.069	1.102
	Average.....		0.904	6.2863	6.1697	0.1166	1.855
Top, center, and bottom	Fines combined, melted and shotted.....	.....	.....	6.722	6.570	0.152	2.261

TABLE V.—*Electrolytic Copper (62 lb.), Melted in Crucible, Gold Added, and Left to Cool Therein and in Furnace.*

Position of Copper in Crucible.	Sample.	Ratio.	Gold. Assay-Method. Fire.	Gold. Assay-Method. Comb.	Gold. Difference.	Gold-Solubility.
			Ounces Per Ton.	Ounces Per Ton.	Ounce Per Ton.	Per Cent.
Top	Top.....	.....	1.929	1.838	0.091	4.717
Center.....	Center.....	.....	1.932	1.804	0.128	6.625
Bottom.....	Bottom.....	.....	1.879	1.756	0.123	6.546
Combined, and separated into fine and coarse.	Fine.....	1	1.852	1.344	0.508	27.760
	Coarse.....	5.460	1.894	1.834	0.060	3.168
	Average.....		1.887	1.758	0.129	6.836

TABLE VI.—*Lead* (99.15 *Per Cent.*) *Melted, Gold Added, Cast into* 9- *by* 9- *by* 9-*In. Cube and Left to Solidify and Cool in a Hot Brick Mold.*

Portion of Cube.	Gold.			Gold-Solubility.
	Assay-Method.		Difference.	
	Fire.	Comb.		
	Ounces Per Ton.	Ounces Per Ton.	Ounce Per Ton.	Per Cent.
Top	1.739	1.707	0.032	1.840
Center	1.777	1.749	0.028	1.576
Bottom	1.715	1.699	0.016	0.933

Alaska Coal-Land Problems.

BY H. FOSTER BAIN, SAN FRANCISCO, CAL.

(New York Meeting, February, 1912.)

[SECRETARY'S NOTE.—This paper, presented in oral abstract at the San Francisco meeting, was not at first supposed by Mr. Bain to be required for publication in the *Transactions;* and the excursion to Japan, in which he took part, immediately after the San Francisco meeting, delayed his presentation of the manuscript. After his return to San Francisco, Mr. Bain, in view of the importance of his paper and its discussion, kindly furnished it in full. The delay in its publication (since it was received in April last) has been due, partly to the circumstance that other contributions had meanwhile acquired the "right of way," and partly to the consideration that, the situation contemplated in Mr. Bain's paper having remained practically unchanged, the paper and its discussion would not lose force or timeliness by this lapse of time.—J. S.]

No acute observer of public affairs can have failed to observe a growing disposition on the part of the public to change emphasis in government from political equality to social justice. When our own government was founded it was felt to be sufficient to secure to each individual the fullest possible freedom of opportunity, and such a scheme was excellently adapted to the rapid development of a large territory having a small population. As the number of people in a given area increases, there are more points of contact and conflict; each must move in a smaller orbit, and there must be more regula-

tion. Germany is making faster progress in industry than is any other country. The principal reason is the insistence there on team-play. In Germany there are fewer multimillionaires and more socialists (spelled with either a big or little "S") than in any other of the leading countries; but Germany wins trade and is steadily improving the conditions of life for her citizens. To some, the change from the freest individualism is welcome; to others it is intensely unacceptable. To this fundamental difference in view-point may be imputed much of the vigor with which the controversy regarding coal-lands has been waged. That present conditions in Alaskan coal-fields are extremely unsatisfactory is generally recognized, but there are wide differences of opinion as to the proper remedies. Large areas of excellent coal within a few miles of tidewater remain unopened, while railroads and other enterprises import coal at a cost of from $7 to $9 per ton on the dock. In building and operating the Copper River & Northwestern railroad to June 30, 1911, British Columbia coal to a total value of $509,943, and Washington coal to the value of $49,438, was imported. The railroad and the leading coast industries have now been converted to burn oil imported from California. In the interior, the price of coal is practically prohibitory, and development of any industry requiring handling of large tonnages is impossible except at the seacoast. Industry in Alaska always will be at a disadvantage, owing to the difficulties and expense of navigation along its coasts, and of transportation into the interior. This, however, is all the more reason why full advantage should be taken of any favorable local feature, such as abundant and excellent coal.

In examining the matter of the opening of the Alaska coal-fields, two distinct questions must be considered. The first, and more pressing, is how to get coal on the market quickly in quantities sufficient to satisfy immediate demands. The second is, what disposition shall ultimately be made of the coal-lands, and under what conditions coal shall be mined in the future. Both questions involve railway problems and are complicated by existing rights of coal-land locators. It is not likely that any satisfactory solution of the second problem will be found immediately. The ramifications of the matter reach such varied interests, and so many things must be taken into account,

that a quick determination might well prove a wrong determination. The first problem, however, that of promptly furnishing Alaskan coal in such quantity as is really needed, seems capable of speedy and safe solution.

It has been estimated by A. H. Brooks that there is a present potential market for 1,000,000 tons of Alaska coal per year. Whether or not these figures be correct, it is undoubtedly true that there is an important market awaiting the opening of the mines. At present all the high-grade coal used on the Pacific coast comes from the East or is imported. The navy, for strategical reasons, uses only the best grade of Pocahontas coal, and when the fleet goes to sea it is followed or preceded by a long string of colliers carrying Eastern coal. At all times there is a procession of boats traveling around South America in order that the battle-ships and cruisers shall have smokeless coal. Necessarily, much coal is burned in bringing this fuel to the ships, and, necessarily, also, the cost of the coal delivered is greatly above that at Eastern tidewater ports, where it has already been raised by the several hundred miles of railway-haul from the mines. In case of war, the supplying of ships in the Pacific with Eastern fuel would be extremely difficult and expensive. Even now the navy is dependent upon foreign tramp steamers. So far it has not seemed wise to convert the whole fleet into oil-burning vessels, because of the difficulty of buying oil in foreign ports. In the opinion of experts, the Bering River and the Matanuska coal-fields of Alaska can furnish a grade of coal excellently adapted to naval use, and the opening of mines there is important in order to afford an adequate and at all times accessible source of supply. The navy uses ordinarily about 150,000 tons of coal per year on the Pacific coast. Possibly the other departments of the government use as much more. In addition, there is a local market in Alaska, now supplied from British Columbia and Washington at considerable expense. Beyond is a strongly competitive market, in which the dominating features are the abundance, low price, and high heating-value of California crude oil. In the three Pacific States together, in 1910, imports and local production of coal amounted to 4,600,000 tons. E. W. Parker has estimated that in the same year fuel-oil displaced 12,000,000 tons of coal in these States. To what extent Alaska coal

can enter this competitive market is uncertain, but the non-competitive market is not unimportant. In it, owing to naval requirements, the government is the largest factor. The obvious thing to do would seem to be for the government to open its own mine and supply its own coal, at the same time meeting the local demands of the non-competitive market by selling the surplus coal at a small advance on the cost of production, as has been suggested by Walter L. Fisher, Secretary of the Interior. The opening of a mine by the government in no way prejudices any future settlement of the larger problem of final disposition of the lands. It will, however, remove any fancied necessity for a hurried decision of the main matter, will relieve the local fuel-situation, will lead to just as much development, and the employment of just as many men, as if some private syndicate undertook the work, and will permit the government finally to act on the larger questions involved with exact knowledge based upon actual experience under the conditions to be met. It has long been customary for large coal-users to maintain a coal-mine, or to have an interest in a coal mining company, so as to have full information regarding costs when making their yearly contracts. For years the Chicago, Milwaukee & St. Paul railroad ran a small coal-mine on this basis, and the information it obtained was worth many times the risk of loss. Why should not the government safeguard the people's interests as carefully as a board of directors for a company looks after those of the stockholders?

Various suggestions have already been made as to how the government should proceed in case it be decided to open a government mine in Alaska. An independent commission has been proposed, but this is, I think, unnecessary and likely to prove expensive. It has also been suggested that the army be placed in charge, and the merit of this lies in the fact that the people have entire confidence in the non-political character of the army and the honesty of its officers. The excellent work, in particular, done at Panama, has greatly raised the popular respect for the Engineer Corps of the army. It is also true that to meet a similar scarcity of coal in the Philippines, the army opened a colliery there and conducted it not only ably, but in such a manner as to demonstrate the value of the coal and of the field, and this led ultimately to the opening of

private mines. The army, however, has other work than mining coal, and no good purpose is served by diverting its officers from their own sphere. Aside from that, the government has in the Bureau of Mines a corps of men especially well qualified to handle this particular matter. In this bureau are all the experienced men necessary to man a colliery, from pit-boss to general manager. Attracted by the opportunity for travel and study, an unusually capable and well-trained lot of engineers has joined the service. A better manager for the proposed enterprise probably could not be found than the chief mining engineer for the Bureau, who has seen entensive service in the coal-fields of Colorado, Iowa, and Illinois, and has served as engineer, superintendent, general superintendent, and consulting engineer for a number of large companies. He has opened, equipped, and run a number of collieries as large or larger than any needed in Alaska, and is thoroughly familiar with coal-mining, not only in every part of the United States, but in England, Germany, and France. There are other men in the Bureau who have had valuable if not as extensive experience, and they are all men of high character, of just the sort to place in charge of a difficult enterprise. Put the matter in their hands with sufficient capital and adequate authority, and the mine will be well and honestly run. Incidentally, it will furnish the Bureau an excellent opportunity to test each new discovery made in its laboratories, and to try out, on a working scale under commercial conditions, its various recommendations for increasing safety and decreasing waste in mining. To the objection that will promptly be made—the high cost of government work—it may be urged that the facts by no means point conclusively to a necessarily higher cost. The Reclamation Service has repeatedly built on force-account at prices below bids from contractors, and the great Los Angeles aqueduct is being built by the City Engineers, and well built, at a cost quite as low as any firm or corporation has ever attained on similar work. On one division the work was done for approximately $2,000,000 less than the lowest bid. Incidentally the Reclamation Service itself has opened and run a government coal-mine, and done it well. Under these conditions it is hardly worth while to answer the old statement that "the government can't run a mine." It can and does. Get good

men and give them reasonable support, and they will build as cheaply for the government as for a corporation. The common notion to the contrary, is based on the excessive cost of jobs let to favored political contractors.

In opening a government coal-mine in Alaska, it is important that conditions be made as nearly as possible identical with those under which a private company would have to work; otherwise one of the large purposes of the enterprise, the obtaining of exact information regarding mining-costs and other conditions, will be defeated. A definite amount of land should be reserved by the Secretary of the Interior, and allotted to the project. Since 2,560 acres is the amount permitted one company under the law of 1908, the same acreage might well be set aside for the government mine. If it is too small for the government, it is likewise too small for a private company, and if large enough for one it is sufficiently large for the other. A definite amount of capital should be allotted to the project, and the coal furnished to the navy and other departments should be credited to the enterprise at the same rate as when sold to individuals. In figuring costs depreciation should be fully taken into account, since the plant will need to be renewed from time to time. A sinking-fund should also be accumulated, and in every particular the enterprise should be run on a business basis.

Since the coal is valueless unless brought to the coast, transportation-problems must enter largely into any solution of the coal-problem. This is true whether the government or some company mines the coal. To reach the Matanuska field, which is favored by Mr. Fisher, it will be necessary to take over and extend the Alaska Central railroad. As this is in the hands of a receiver, its owners would probably welcome the opportunity to sell to the government. The most difficult part of the road is already built, though much money would be needed to put it into good condition. Owing to grades and curves between Seward and Knik, the line would not be easily or economically operated; and large tonnages over this division would be expensive to handle. It is proposed to establish a summer harbor at Knik, and to ship the bulk of coal from this point, to which railway-transportation is easy. The line to Seward could handle quick freight and afford an emergency-

route for coal. Mr. Fisher has recently suggested that the men and equipment soon to be out of commission at Panama be transferred to Alaska, and employed in building the proposed trunk-line from Seward to Fairbanks. There is much to commend this suggestion, since, under the plan proposed, a tried and effective force would be made available for the work.

As to the need of a trunk-line to the interior of Alaska there can be no question, if the country is ever to be developed. The practical question is wholly as to means and method. Into the Bering River coal-field, the one most easily accessible, a number of surveys have been made. There are, however, only two routes that need to be considered. The first is a direct line approximately 25 miles long to Controller bay or Katalla. This requires the making of a harbor, and any estimate of expense of shipment by this route must take this into account. Eventually one or more such lines is likely to be built, but for the present the cheapest route would seem to be the longer one to Cordova. This requires a road from the mines almost to Katalla, then north to a junction with the Copper River & Northwestern railroad. The distance is longer, 58 miles, but of the total, 33 miles form part of the main line of the road already built, and the cost of an extra harbor is avoided. This is the more feasible route for the present. It would have the advantage of permitting through shipment to the copper-mines of the interior, while the other would require transfer to boats at Katalla and retransfer to cars at Cordova. A considerable part of the local benefit to be derived from opening the mines would be lost if connection were not made with the Copper River & Northwestern. The tonnage that would be furnished by the government's own coal-requirements would probably be sufficiently attractive to cause one or the other of these roads to be built by private capital in case the decision favors the Bering River field. Wherever the work be started, the Secretary of the Interior or the Director of the Bureau of Mines should be authorized to negotiate for transportation to the sea on a basis of an agreed annual tonnage determined by naval requirements, and fixed rates reviewed by the Interstate Commerce Commission. If, on these terms, private capital be not forthcoming for a railway-line, the latter should be built by the government and operated directly or through lease, as

might be determined at the time. The cost of the road to the coal-fields would be less than that of colliers to bring coal to the navy from the East, and the cost of operating the whole mining enterprise, granted that the worst estimates of the excess-cost of government work prove true, would be less than the excess-price now paid on the Pacific coast for Eastern coal. It is taken for granted that arrangements would be made for the navy and other government departments to use the government coal on the Pacific coast. It would probably be necessary to establish supply-stations at the tidewater terminals, on Puget sound, and perhaps elsewhere, and to deliver coal to them either by contract or by using government colliers. If any coal be sold outside Alaska (and as to the advisability as well as possibility of that there would be much question), the cost of transportation should of course be added to that of mining.

Whatever be the decision as to government ownership and operation, leasing, or any other method of opening the Alaskan fields, existing rights should be adjudicated as promptly as possible. Under the terms of the Cunningham decision, if they be adhered to, it seems doubtful whether many, if any, of the claims will ever go to patent. In any event, final decision cannot be expected promptly; and what is needed now is the prompt opening of one or more mines. If any of the existing claims are valid they should be promptly passed, since the opening of a private mine or two would supplement rather than conflict with the government colliery, and would permit the latter to be devoted practically to furnishing coal for the navy and other government departments. The potential competition of the government mine would prevent any unreasonable increase in local prices, even in the event that actual competition of oil did not operate to do so.

The adoption of the plan outlined would (1) make the navy independent as to fuel-supply in the Pacific; (2) furnish coal at reasonable cost to local industries needing it; (3) permit the government to determine the exact facts as to costs and conditions of mining; (4) give the Bureau of Mines an excellent field for testing its theories and recommendations; (5) permit adequate time to be taken to adjudicate existing claims, and formulate a really satisfactory policy of handling the coal-lands of Alaska. Adoption of the plan outlined would not dispose

of the main question, the final disposition of Alaskan coal-lands. I am by no means sure that a satisfactory final solution can now be formulated; and until it is clear that the right method has been found, I, personally, would favor leasing whatever acreage may be needed for the fullest development, upon terms outlined later. It should be remembered that the United States is the owner of these and other public lands as trustee for the whole people, and Congress is the sole judge of the terms under which they may be occupied or sold. Whatever, in the judgment of Congress, is for the greatest benefit of the people of the United States, not Alaska alone or the Pacific coast, is the right and proper thing to enact into law, and what has been wisest in the past may very well not be, in the judgment of Congress, now equally wise.

In discussing the Alaska coal-land situation it is commonly assumed that the matter is primarily one of conservation. This is far from being true. Any analysis of the speeches that have been made, and of the papers written on the subject, will make clear that distribution of profit rather than waste of coal is what the proponents of change have chiefly in mind. Engineers, on the other hand, are primarily concerned with development, and as conservationists their interest centers mainly upon securing development with the minimum of waste. As has been repeatedly pointed out, development can be expected only if some profit accrue to, or be anticipated by, the developers. Avoidance of waste is a matter of securing the best technical service in the actual conduct of an industry. Economic considerations determine the conditions under which development may or must proceed; the engineer determines how best to meet a given set of conditions. Any circumstance which permits more money to be spent on a given process or a larger control of an industry to be exercised, makes possible a greater saving of material. Higher prices and larger units of business organization make for a less waste of coal or any other material. A single syndicate mining and selling Alaska coal would undoubtedly conserve both coal and capital better than any other organization. To turn the matter over to such a syndicate would be the simple and business-like method of procedure—if these were the only considerations. It is evident, however, that such a bald solution of the problem would

satisfy no one except the members of the syndicate, and some other method or some modification of this one must be found if progress is to be made.

Recurring to the statement, then, that as a people we are moving away from our early intense individualism, it is clear that there is now a widespread desire to protect the average citizen from extortion following monopoly, and to prevent the accruing to individuals of the unearned increment on mineral lands. The problem, as thus stated, is world-wide and it is to be faced in every industry. Its solution as regards the existing public mineral lands would be but an incomplete answer to the great needs of present civilization—(1) the realization of the benefits of large business organizations without loss of control of monopoly; (2) the equitable distribution of the surplus earnings of mankind. Solution of the Alaskan coal-problem will be at best but one small step in the solution of the general problem, and any method applied in Alaska and based upon government ownership of the land can be of but small immediate importance, since for many years to come the people of the United States will draw the bulk of their fuel from fields that have already passed into private ownership. It is true that approximately one-third of the known coal is still owned by the general government. This, however, is undeveloped. The practical question is, whether it can be opened under regulations that will rectify mistakes flowing from past policies, or at least prevent similar mistakes in the future.

The great anthracite-deposits of Pennsylvania are already in the hands of a monopoly, and in a large measure the enhanced value of the land has already gone to the individuals concerned in organizing and maintaining it. There is no question but that the industry has been benefited, but there is widespread popular dissatisfaction with the control exercised, in this case by private individuals, and the government finds itself estopped from effective regulation of the industry by the fact that the lands are owned privately. Recent efforts of the government to enforce the Sherman Act, while nominally successful, seem to have proved barren of actual result. In the bituminous coal-mining industry, on the other hand, where intense competition is the rule, conditions are chaotic and intensely unsatisfactory to every one except those large users of steam coal who put low

price above every other consideration. The average citizen who buys in small quantities is already paying high prices. By stringent anti-monopoly laws, the tendency towards monopoly was halted before the bituminous coal-mining industry became completely organized. It would, however, be extremely difficult to reach a fair and disinterested conclusion as to whether, from the point of view of the general public, present conditions are more unsatisfactory in the anthracite or the bituminous fields. Low prices do not tell the whole tale. Wasted capital, wasted coal, low wages, and lost lives, the formation of a large class of permanently discontented workmen—all these must be counted against low prices of coal used by railroads and power companies, granted even that low prices to the latter mean some benefit to the ultimate consumer of transportation and power.

An instructive lesson may be drawn from the anthracite-fields. Many persons forget that for years coal was mined and sold there under fiercely competitive conditions. At a time when it cost but $1,000 to open a mine, and $10,000 sufficed "for quite a respectable business," many rushed into the industry. Overproduction worked the inevitable result of low prices and no profit. Since, however, there was a profit to the railroads in transporting the coal to the market, mining was stimulated by loans from the railroads to the struggling coal companies. Eventually the railroads had to take the mines to protect their loans, and then discrimination in rates and service became inevitable. In time the cost of opening mines rose to almost prohibitive figures, and no one could afford to invest the large sum necessary unless he also controlled transportation. All these factors worked together to produce the inevitable result, a monopoly of anthracite-mining in the hands of the railroads. From many points of view the results have been excellent. The properties, however, are operated, not for the sake of the mines, but for the railroads, and a great national industry is subordinated to the immediate necessities of a group of roads. For many years there was no profit in the actual mining of anthracite; probably there is little now. Millions have been paid in royalties to fortunate owners, more millions have been made in the business of transporting and marketing the coal; but in the business of mining there has been no con-

siderable profit. There has always been a surplus of mines and of railroads in the anthracite-fields, and, the larger capital being invested in the railroads, it was inevitable that they came to dominate the situation. The State and the National legislators never aided the industry by constructive legislation based upon any real consideration of the larger problems of the people and the coal. When they acted they merely pecked at the industry with petty restrictive acts. The net result is a firmly-seated private monopoly of a great national resource.

In the bituminous coal-fields the story is similar to that of the anthracite-fields in the early years—an abundance of coal and of capital, investments made on the basis of an inflated industry during "boom" times, overproduction, idle plants and men, wasted money and coal, attempts of the railroads to control production, each in its own interest, and the formation of monopolies in limited areas. The tendency towards a larger monopoly has been checked by anti-monopoly legislation and the regulation of interstate commerce. In the meantime many idle colliery-plants stand gaunt and rusting in the valleys and on the prairies, and an idle plant does no service for any man. It is useful only as a club in considerations of rates and prices. Clubs, it is well to remember, are the weapons of the stone age.

The two industries exemplify the effects of combination on the one hand, and of competition on the other. In 1891, under competitive conditions, anthracite sold at the mine for an average price of $1.73 per ton. In 1910, under monopoly conditions, the price had risen to $2 per ton, an increase of 16 per cent. In the same years bituminous coal, under competitive conditions, rose in price from $0.99 to $1.12 per ton, an increase of 13 per cent. These figures are no measure of retail prices, but it is evident that whatever may have been the effect as regards consumers, the creation of a monopoly in the anthracite-fields has benefited producers of coal more in other directions than by increase in price. It is important to remember in this connection that the cost of selling and delivery is a large element in the retail price of coal, as of other products, and often the buyer of small lots imputes to the original producer charges that originate long after the coal reaches the top of the ground.

It seems clear, on the whole, when the two industries are contrasted, that combination rather than competition yields better results. To secure its full benefits without injury to the public is the real problem, and some form of public control is evidently desirable. The simplest form of control yet found is ownership. It does not follow that public ownership of all the coal-mines is either desirable or necessary, but, so long as the government already owns one-third of the coal-reserve of the nation, this coal should not be allowed to pass into private hands until some adequate means of meeting the problems outlined has been found and tested. It has repeatedly been proved that ownership of a majority of the stock is not necessary in order to control a corporation, and a holding of one-third of the coal is so much greater than any possible combination of lands privately owned, that, particularly in combination with its other powers, the government can determine absolutely the future of the coal-business of this country. The government ought not to sell the coal-lands until the whole matter is settled. In the meantime, all the demands for development can be amply met by a simple leasing-system, such as has been proposed by the administration. There is no benefit to be derived from locking up the coal-lands in either government or private hands, but to sell them irrevocably, at nominal prices and in quantities in excess of the needs of present development, is on a par with the giving of perpetual franchise for municipal utilities. The leasing-system may or may not prove the correct procedure as a permanent policy, but for the present it has many advantages over any other.

The problems that now confront our bituminous coal-mining industry arose a few years ago in the same form in Germany. In that land of orderly industry, the coal business was in as bad condition as was ever that of Pennsylvania or any other American State. There was this difference: the Prussian government, as the owner of many miles of railroad, was concerned, in place of, as in America, a group of private individuals; but, further, the government there did not, as here, own a large part of the known coal. Difficulties in railroad-operation, as well as the general disorganization of industry, focused public attention on the coal industry. The State railways sometimes enjoyed cheap coal, and at other times paid dearly for it. At

times the lines had much traffic, and at other times little. For short periods the collieries made large profits, and these periods were succeeded by long lean years. The methodical Germans studied the situation, and then met it in a practical manner, undeterred by theoretical considerations of what might happen. The government purchased enough mines to give it control of approximately 40 per cent. of the coal-producing capacity of the country. Then, on the basis of the ownership of these mines and a thorough knowledge of the business, the government joined in a syndicate for mining and marketing coal. While there has been some criticism of the results, there has been steady work, moderate but regular profits, and better prices. English coal companies, it is true, have made inroads on the Berlin market, but the Germans gaze with equanimity upon this. They regard the low-priced coal furnished by competing English companies as just so much gain. Germany has regulated the coal-monopoly by becoming a party to it.

I am among those who believe that "trusts," meaning thereby large business organizations, are inevitable. The conditions of modern industry favor them, and their economic advantages outweigh their evils. It is not necessary, however, to allow them to proceed unregulated, and to do so is as foolish as to allow unrestricted employment of any other new device, such as automobiles or wireless telegraphy. Trusts need regulation, but if business is to be conserved, it must be by informed and sympathetic regulation. The best way to control a trust is through the board of directors. Even a minority there will be listened to with respect, and suggestions, if sensible, will be adopted.

In Alaska the conditions of industry are naturally adverse. It is extremely difficult to develop an enterprise successfully, and only the most efficient form of organization can hope for more than sporadic success. The coal-fields may be developed more quickly, economically, and efficiently by a single great syndicate than by a number of small warring concerns. The first step in the solution of the Alaskan coal-land problems should be the opening of a government mine, primarily to supply the government needs of coal. If necessary, the government should build the needed railway to the mine. The second step should be the opening of private collieries on leased

ground, and the formation of a selling syndicate to pro-rate all contracts. The government itself should be a member of this syndicate and should be represented in its management by an expert resident official of ample authority. If any of the present claimants of lands succeed in getting patents, they should be invited to become participants in the syndicate, and they would probably find it greatly to their advantage to join.

The immediate thing is to provide for mining some coal now and still leave the policy of the future to be shaped to meet its own needs. This seems to me to be most easily accomplished by following the opening of the government mine with an offer of a limited number of leases to any one who may care to undertake the risks involved in opening collieries in Alaska. These risks, from a business point of view, are bound to be large for many years, and the government in offering a lease should be prepared to safe-guard in every reasonable way the lessees as well as the public.

Leases should be granted upon liberal terms, fixed in advance, should run for a period of preferably 25 years, and should cover in each case sufficient area to permit continuous operation of a modern plant through the life of the lease. The Mining and Metallurgical Society of America has suggested a lease for all the coal in the ground, with a minimum required production that rapidly increases with acreage, in order to afford security and yet not permit holding ground for speculation. There are many possible ways of meeting this problem, and any one of several will prove satisfactory, provided it be clearly announced in advance and remain unchanged during the operations of the lease. The royalty should be low, since a healthy industry is more important than revenue to the government, and it is no proper function of the government to make money out of one industry or class of citizens to the profit of others. The number of leases operated at any one time should be restricted to correspond to the probable market for coal. Whenever, however, the existing collieries become unable to supply the demand, or the profit on the business for two successive seasons amounts to more than a fixed percentage on the capital invested, due allowance having been made for maintenance and sinking-fund, the government should offer additional blocks of ground for lease, and the new collieries, when open, should be-

come members of the selling syndicate and should receive their *pro rata* of contracts. Leases had best be let by competitive bidding. By this method the companies would fix their own royalties. In the initial competition they should be allowed to offer alternative bids based upon the number of leases that it may be decided to let. Clearly, a company having the whole business can pay a higher royalty than one which must divide the trade with others. It is true that competitive bidding has disadvantages as compared with assignment by lot or in order of application, but among those familiar with the business and having the capital to carry out the terms of the lease, bidding would be equal, and there can be no advantage in having others undertake to open collieries.

All applications for leases should be accompanied by bond, and after two years within which to open the mine, royalties should be collected for the proportionate tonnage of each year whether coal be mined or not. Such royalties should apply on future production, their payment being merely required to prevent indefinite suspension of operation. The detailed terms of the suggested leases might profitably be varied somewhat, but the following general principles should be held to: (1) Terms liberal and known in advance; (2) all companies to participate in sales and contracts; (3) the government to be fully informed of all details; (4) the government to be in position to prevent extortion by permitting the opening of additional mines; (5) royalties to be determined by the companies themselves by means of competitive bidding; (6) revenue over and above the cost of maintaining the service should go in large part, if not entirely, to the local Territorial government; (7) leases should be freely transferable, and subject to cancellation only for material failure to observe their terms and after court review.

Until the California oil-fields begin to show signs of depletion, it is not likely that development of an Alaska coal-mining industry will be rapid, but under the plan suggested above, every opportunity would be afforded for opening mines as fast as the market could be developed, and that without sacrifice of any interest, present or future. Such a complete change in policy should not be attempted on too large a scale at once; and for that reason it is suggested that the plan be applied to

Alaska only at present. If the results prove satisfactory the same system can be extended to the remaining coal-lands in the States, and there can be little question that a selling syndicate of this sort could bring to the bituminous coal industry all the benefits now apparent in the anthracite-region, without sacrifice of public interests. The advantage of the leasing-system is that it does not unduly commit the future of the industry, and as new leases came to be given from time to time the companies would offer higher royalties if the business warranted, or the price of coal would fall. In either event the "unearned increment" would accrue to the public rather than to private individuals. The plan does away with speculation in the land, which is exactly what it is designed to accomplish. It limits the amount of speculation possible in leases by fixing a maximum profit that may be realized. By way of compensation it also limits the chances of loss by restricting competition, and by throwing on the government much of the expense of pioneering, in that the government mine, to be opened first, must have transportation-facilities, and later comers may enjoy these equally with the government. It meets or attempts to meet the Western objection to leases, namely, government from Washington, and constant interference as to details, by requiring the terms of the lease to be fixed in advance and that leases shall only be canceled in the same manner and on the same grounds as are private contracts. A further important feature of the plan is a resident official with ample authority to exercise the somewhat limited discretion proposed. No attempt is made to fix prices. These will be determined by competition of Canadian and American coal, California oil, and the cost of hydro-electric power. The limitation is as to profits, and that is to be enforced in the normal method by bringing in additional competition when profits become too large. An orderly rather than speculative method of setting this force into play is provided. The official whose duty it will be to offer new leases will be fully informed as to the facts and must proceed according to fixed conditions known in advance. Finally, his judgment will be put to test, since, unless the showing of large profits or insufficient supply be satisfactory, no competent bidder will be found when the leases are offered. In the meantime the producers are protected against cut-throat

competition among themselves, the power of the government is available to compel fair transportation-rates, and the selling of the product, the most difficult and delicate part of the coal business, is in the hands of the companies and not of the government. The plan provides for placing the producing companies upon the best economic basis that competition of other fields will permit, and under these conditions they will be prepared to mine coal according to the best practice that is now possible. Conservation of coal and protection of life in the mines must be enforced through the police power of the State and regulations directed to that end must be matters of law. In order to secure good regulations and their vigorous enforcement, however, the operating companies must be placed in a satisfactory financial position. This is only possible when they are protected against unfair competition and unreasonable capitalization.

It is too much to assume that any plan will prove entirely satisfactory, but the above is offered as one which at least takes account of the difficulties and provides means of meeting those that can be foreseen.

Flameless Combustion.

BY CARLETON ELLIS, MONTCLAIR, N. J.

(Presented at a meeting of the New York Local Section of the Institute, April, 1912.)

I. Introduction.

The problem of the influence of hot surfaces upon gaseous combustion is one which, from a purely scientific standpoint, has engaged, for many years past, the attention of Prof. William A. Bone, of Leeds University and the Imperial College, London; and as his recent work has been the direct outcome of earlier scientific investigation, it will be appropriate, by way of introduction, to review briefly the present position of science with respect to this important subject, as stated by Professor Bone..

One may perhaps best arrive at an understanding of the term "flameless" or "surface" combustion by considering certain facts which differentiate it from the more familiar processes of

combustion as they occur in ordinary flames. All hot surfaces have an accelerating influence upon chemical changes in gaseous systems. If, at any temperature, a gaseous system, *A*, tends to pass over into another system, *B*, contact with a solid at the same temperature will accelerate the process.

To take a very simple example, if a mixture of hydrogen and oxygen in their combining proportions (electrolytic gas) were maintained in an inclosure with smooth glass walls at a temperature of, say, 450° C., there would certainly be a tendency to form steam, but the rate of change would be negligibly small. If, however, there were brought into the system some porous solid material at the same temperature, so that a large surface was exposed to the gases, the rate of change would at once be rapidly accelerated in the layer of gas immediately in contact with the hot surface. Steam, the product, would diffuse outward from the surface, and the supplies of hydrogen and oxygen at the surface would be renewed by diffusion inward. Thus combustion would proceed heterogeneously at the surface until the transformation of the original electrolytic gas into steam was complete. In the circumstances just cited, the rate of combustion, although now quite measurable, would probably be insufficient to cause any self-heating of the inclosure. The temperature would remain at 450° C., which is well below the ignition-temperature of the combustible mixture, or the point at which a solid would attain even incipient incandescence.

It is therefore necessary to distinguish between two possible conditions under which gaseous combustion may occur, namely: (1) homogeneously, that is to say, equally throughout the system as a whole, at temperatures below the ignition-point, slowly and without flame, and at temperatures above the ignition-point, rapidly and with flame; and (2) heterogeneously, or only in layers immediately in contact with an incandescent surface ("surface" or "flameless" combustion). Other things being equal, the heterogeneous surface-combustion is a faster process than the normal homogeneous combustion of ordinary flames.

The influence of hot surfaces upon combustion at low temperatures seems to have occupied the attention of several chemists (Dulong and Thenard and, independently, Dobereiner, in France, Sir Humphry Davy, William Henry, Thomas Graham,

Faraday, and de la Rive, in England) during the first third of the last century; but no one of these distinguished men succeeded in evolving a satisfactory theory of the phenomenon, nor, with the exception of the famous "Dobereiner lamp," was there any practical outcome of their efforts. In 1836, after a long but abortive controversy between Faraday and de la Rive, interest in the subject was dropped, not to be revived until recent years. Professor Bone's attention was first drawn to the subject during the course of an investigation on the combustion of hydrocarbons at low temperatures. The subject soon became so absorbingly attractive that he embarked upon what proved to be a long inquiry into the influence of a great variety of hot surfaces upon the combination of hydrogen and oxygen at temperatures below the ignition-point. The inquiry has also included other cases of slow combustion; and experiments now in progress in his laboratory will materially advance the science of the subject.

Professor Bone's experimental results justify the conclusion that the power of accelerating gaseous combustion at temperatures below the ignition-point is possessed by all surfaces in varying degrees, dependent upon their chemical characters and physical texture. Moreover, the "activity" of a given surface can be enhanced or diminished at will in a truly marvelous manner by previous special treatment. Thus, for example, in the case of the combination of either hydrogen or carbon monoxide with oxygen, in contact with a non-oxidizable metal or non-reducible oxide, the "activity" of the surface may be greatly stimulated by previous contact with the combustible gas, and, conversely, may be diminished by previous contact with oxygen. Again, there is abundant evidence that the actual surface-combustion is dependent upon a prior "absorption" (or condensation) of the combustible gas, and possibly also of the oxygen, by the surface. To what extent oxygen is involved, is not as yet perfectly clear. The "absorbed" (or condensed) gas becomes "activated" (probably "ionized," as the physicists would call it) by association with the surface. Finally, certain important differences have been established between ordinary homogeneous combustion and heterogeneous surface- or flameless combustion. Thus, for example, whereas the presence of water-vapor certainly accelerates, if it is not essential to, the

homogeneous combustion of carbon monoxide, it greatly retards the heterogeneous combustion of the same gas in contact with a surface such as fire-clay. Again, whereas methane has, in ordinary flames, a much greater affinity for oxygen than either hydrogen or carbon monoxide, a hot surface, by virtue of some "selective" action, will completely reverse this usual order of things—a remarkable circumstance, than which no better proof could be afforded of the reality of surface-combustion.

In a discussion before the British Association in 1910, Sir J. J. Thompson insisted that combustion is concerned not only with atoms and molecules, but also with electrons—*i. e.*, bodies of much smaller dimensions and moving with very high velocities—and suggested that "in reference to the influences of hot surfaces in promoting combustion, to which Professor Bone has drawn attention, it was not improbable that the emission of charged particles from the surface was a factor of primary importance." Those who have followed recent developments of the corpuscular theories of electrical action will recall the experimental proof that incandescent surfaces emit enormous streams of electrons, traveling with high velocities; and the action of such surfaces in promoting combustion may ultimately be found to depend on the fact that they bring about the formation of layers of electrified gas, in which chemical changes proceed with extraordinary rapidity.

A distinguishing feature of the new processes employing flameless combustion is, that a homogeneous explosive mixture of gas and air, in the proper proportions for complete combustion (or with air in slight excess), is caused to burn without flame in contact with a granular incandescent solid, whereby a large proportion of the potential energy of the gas is immediately converted into radiant form. The advantages claimed for the new system are: (1) the combustion is greatly accelerated by the incandescent surface, and may be concentrated just where the heat is required; (2) the combustion is perfect with a minimum excess of air; (3) the attainment of very high temperatures is possible without the aid of elaborate "regenerative" devices; and (4), by reason of the large amount of radiant energy developed, transmission of heat from the seat of combustion to the object to be heated is very rapid. These advantages are so uniquely combined in the new system that

the resultant heating-effect is, for many important purposes, not only pre-eminently economical, but also easy of control.

II. Diaphragm-Heating and Its Applications.

In this process the homogeneous mixture of gas and air is allowed to flow under slight pressure from a suitable feeding-chamber, through a porous diaphragm of refractory material, and to burn without flame at the surface of exit, which is thereby maintained in a state of red-hot incandescence. The diaphragm is composed of granules of fire-brick bound together into a coherent block by suitable means. Its porosity is graded to suit the particular kind of gas for which it is to be used; but for undiluted coal-gas, or coal-gas containing only a small proportion of water-gas, a diaphragm so porous that the gaseous mixture will readily flow through it at a pressure of ⅛ in. water-gauge is employed. It is mounted in a suitable casing; the space inclosed between the back of the casing and the diaphragm constituting a convenient feeding-chamber for the gaseous mixture, which is introduced at the back. Such a mixture may be obtained in either of two ways: namely, (1) by means of suitable connections through a Y-piece with separate supplies of low-pressure gas and air (2 or 3 in. water-gauge only is sufficient); or (2) by means of an injector-arrangement connected with a supply of gas at 2 lb. per sq. in. pressure. In this case the gas draws its own air from the atmosphere in sufficient quantity for complete combustion; the proportions of gas and air being easily regulated by a simple device.

To start up a diaphragm, gas is first of all turned on and ignited as it issues at the surface; air is then added gradually until a fully aerated mixture is obtained. The flame soon becomes non-luminous and diminishes in size; a moment later, it retreats to the surface of the diaphragm, which at once assumes a bluish appearance; soon, however, the granules at the surface attain an incipient red heat, producing a curious mottled effect; and, finally, the whole of the surface-layer of granules becomes red-hot and an accelerated "flameless combustion" comes into play. All signs of flame disappear, and there remains an intensely glowing surface—a veritable wall

of fire but without flame—throwing out a radiant heat which can steadily be maintained as long as required.

The actual combustion in the diaphragm is confined within a very thin layer ($\frac{1}{8}$ to $\frac{1}{4}$ in. only) immediately below the surface, and no heat is developed in any other part of the apparatus. While the front of the diaphragm is intensely hot, the back of the apparatus is so cold that one can lay the hand on it. The combustion of the gas, although confined within narrow limits, is perfect; for, when once the relative proportions of gas and air have been properly adjusted, no trace of unburnt gas escapes from the surface. Moreover, the temperature at the surface of the diaphragm can be instantly varied at will by altering the rate of feeding of the gaseous mixture; there is no lag in the temperature-response—a circumstance of great importance in operations where a fine regulation of heat is required. The temperature of a diaphragm working on a mixture of coal-gas and air, at a given rate of feeding, depends on whether or not the intense radiation from its surface is impeded; with a freely radiating surface, the temperature of a properly-made diaphragm may be maintained at any point up to about 850° C. (say 1,550° F.), according to the rate of supply of combustible mixture. A curious feature of the diaphragm is the freedom from back-firing at this or lower temperature. Even when an explosive gaseous mixture is passed through the porous wall at a velocity very much smaller than the normal speed of back-firing of the mixture, no explosion backward will occur. Such a plain diaphragm may be placed at any desired angle between the horizontal and vertical planes.

The diaphragm method is appliable to a variety of combustible gases. Coal- or coke-oven gas (either undiluted, or mixed with water-gas), natural gas, gasolene-air gas, carburetted water-gas, are all well suited in cases where unimpeded radiation is required. I have recently found compressed liquefied gas (Blau gas) to give satisfactory results. Also, Professor Bone has constructed and successfully operated plane diaphragms of all sizes up to 4 sq. ft. in area, and is able to vouch from experience that their durability and radiant power are unimpaired, even after long-continued use.

Incandescence Not Dependent on External Atmosphere.

A further important point with regard to diaphragm-heating is, that the incandescence of the surface in no way depends upon the external atmosphere. When once the diaphragm has become incandescent, and the proportions of air and gas supplied in the mixing-chamber at the back have been properly adjusted, the surface will maintain its incandescence unimpaired even in an atmosphere of carbon dioxide, nitrogen, or steam.

Applications of Diaphragm-Heating.

I need hardly point out the many obvious purposes to which "diaphragm-heating" may be applied. Broiling, roasting, toasting, are at once suggested; others will doubtless occur to you;—such efficient means of attaining radiant heat can hardly fail to find new industrial uses. The evaporation and concentration of liquids by means of radiant energy emitted from a diaphragm fixed in a horizontal plane above the surface of the liquid is readily carried out.

For example, the evaporation of a solution of sodium silicate (water-glass) is an operation which could not be satisfactorily performed by the ordinary means of heating the vessel by flame from below. By the new method, however, only the topmost layers of the liquid are heated; the radiant energy of the diaphragm is instantly transmitted to the surface of the liquid, where it is absorbed and utilized for the evaporation. The sodium silicate separates out as a skin on the surface of the liquid, it is then dried by the radiant heat, and at intervals the crust of dry sodium silicate may be skimmed off. In this way, we are not only able to evaporate the solution with a great economy of heat, but we are also able to complete the evaporation of highly concentrated solution much more easily than by means of heat applied from below.

III. Incandescent Surface-Combustion in a Bed of Refractory Granular Material.

This process is applicable to all kinds of gaseous or vaporized fuels, and to a great variety of both small- and large-scale industrial heating purposes. It consists essentially in injecting through a suitable orifice, at a speed greater than the velocity

of back-firing, an explosive mixture of gas (or vapor) and air, in their combining proportions, into a bed of incandescent granular refractory material, which is disposed around or in proximity to the body to be heated.

Fig. 1 shows the process as applied to the crucible-furnace. The crucible is surrounded by a bed of highly refractory granular material. The mixture of gas and air is injected at a high velocity through a narrow orifice in the base of the furnace, and as it impinges upon the incandescent bed, combustion is instantaneously completed without flame.

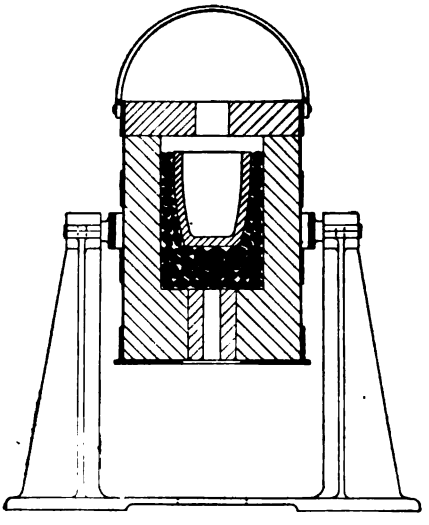

FIG. 1.—CRUCIBLE-FURNACE HEATED BY SURFACE-COMBUSTION.

The seat of this flameless combustion is in the lowest part of the bed; and the burnt gases, rising through the upper layers, rapidly impart their heat to the bed, maintaining in it a high degree of incandescence. Fig. 2 shows a similar arrangement for the heating of a muffle-furnace, an arrangement which needs no further explanation.

It is obvious that this process is adaptable to many other furnace-operations, as, for example, the heating of retorts, annealing-furnaces, and the like. It is not essential that the bed of refractory material shall be disposed around the vessel or chamber to be heated; it may equally well be packed into tubes, or the like, traversing the substance to be heated. This

latter modification is important in relation to the melting of metals or alloys which are fusible at temperatures below about 600° C., and also in relation to steam-raising in multitubular boilers. By this process, much higher temperatures are attainable with a given gas than by the ordinary methods of flame combustion without a regenerative system. In fact, we have found that with any gas of high calorific intensity (such as coal-gas, water-gas, or natural gas), the upper practicable temperature-limit is determined by the refractoriness of the material composing the chamber (*i. e.*, the muffle or crucible) to be heated, rather than by the possibilities of the combustion itself.

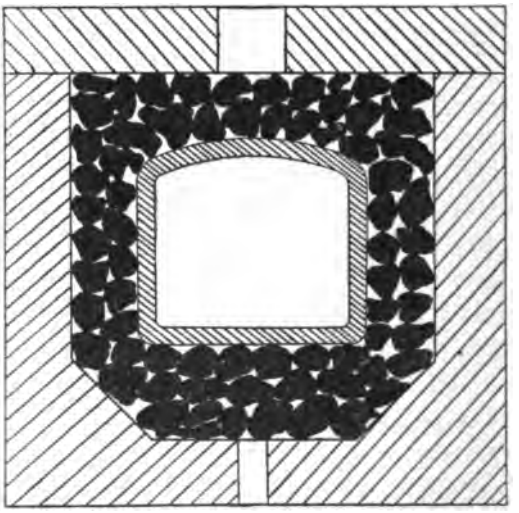

FIG. 2.—SURFACE-COMBUSTION APPLIED TO A MUFFLE-FURNACE.

In a crucible fired by coal-gas on this system we have readily melted Seger cone No. 39, which, according to the latest determination of the *Reichsanstalt* in Berlin, melts at 1,880° C. We can also easily melt platinum, showing the possibilities of the method in regard to high temperatures with gas-fired furnaces. Using air preheated to 500° C., with coal-gas, a temperature estimated at somewhat over 2,000° C. has been attained. A very resistant chromite, not melting at 1,880° C., was fused in this way. Crucibles of alundum are fused without preheating the air.

For the very high temperatures obtained with coal-gas, water-gas, or natural gas, Professor Bone employs a bed composed either of fragments of magnesia, which has been burned at a high temperature, or of a neutral and highly refractory material specially prepared for this purpose. When the temperature required does not exceed 1,300° C., the bed of refractory material may be composed of a good quality of fire-brick, crushed and meshed to a suitable size. As already remarked, the method is applicable to all kinds of gaseous and vaporous fuels, but naturally the maximum temperature obtainable in any given case will depend upon the volume and heat-capacity of the products for a given heat-development in the bed. Thus, while with actual coal-gas, water-gas, or natural gas, it is possible to attain temperatures up to at least 2,000° C., about 1,500° C. would probably be the maximum temperature obtainable without regeneration with producer-gas of low calorific intensity, such as Mond gas. With some degree of heat-recuperation, which in such a case would be quite practicable, this limit could be in all probability considerably exceeded.

The following are the results of a test on a muffle-furnace in which the muffle was heated between 815° and 1,425° C., with fully aerated coal-gas.

Results of Test on a Muffle-Furnace.

Dimensions of muffle, 9.5 by 5.25 by 3.25 in.

Temperature in Middle of Muffle.		Gas-Consumption to Maintain Temperature Constant.	Temperature of Product.	
Degrees C.	Degrees F.	Cubic Feet Per Hour, at 15°C.	Degrees C.	Degrees F.
815	1,499	21.0	540	1,004
1,004	1,840	35.3	645	1,195
1,205	2,201	58.0	870	1,598
1,424	2,596	79.0	1,085	1,985

Mean net calorific value of gas = 540 B.t.u. per cu. ft. at 15° C.

The conditions under which the tests were carried out made possible the accurate determination of the rate of gas-consumption requisite to maintain the muffle at any constant temperature between 815° and 1,425° C.

The temperatures given in the first two columns are those recorded by a standard thermo-junction placed in the middle of the muffle. The temperatures of the escaping products were also ascertained by means of a standard thermo-junction. It will be observed that the temperature of the products is in

every case some 300° or 400° C. lower than that of the muffle. Even with a muffle-temperature of 1,424° C., there was no appearance of flame whatever at the top of the furnace. The gas-consumptions recorded in the middle column are extremely economical in comparison with ordinary heating by flame-contact. Thus, for example, in a similar test with a muffle of the same size, heated by flame-contact in a furnace of modern design, the gas-consumption to maintain the muffle at 1,055° C. (the maximum temperature obtainable) was 105 cu. ft. per hour; whereas by interpolation in the above table the consumption in the surface-combustion furnace at the same temperature would have been about 43 cu. ft. per hour only.

In a test which I witnessed, the consumption of gas by the Bone muffle-furnace was just one-half of that of a good type of the ordinary muffle-furnace of similar capacity, both being maintained for several hours at 1,500° C.

IV. Surface-Combustion as Applied to Steam-Raising in Multitubular Boilers.

It is well known that hitherto the gas-firing of steam-boilers has not been very successful, either in thermal efficiency or in the rate of evaporation. All the gases used in America for raising steam, such as blast-furnace gas, the surplus gas from by-product coke-ovens, natural gas, and producer-gas of various compositions, have been found amenable to the surface-combustion system. An eminent English blast-furnace engineer estimates that the efficiency of the best type of water-tube boiler, fired by blast-furnace gas, does not exceed about 55 per cent. Professor Bone asserts that careful observations, made on a battery of Lancashire boilers, fired by blast-furnace gas, evaporating water previously softened to within 4 degrees of hardness, with an attachment of the most approved type of economizers (so that the temperature of the burnt gases going to the chimney was reduced to the lowest possible point consistent with good draft), proved that the thermal efficiency did not, under the best of conditions, exceed 60 per cent. For boilers fired by coke-oven gas, one can safely say that the average thermal efficiency does not exceed 65 per cent., while in exceptional cases it may amount to perhaps 70 per cent.

Fig. 3 represents a multitubular boiler of cylindrical section, operated by flameless combustion. It is traversed horizontally by a series of steel tubes, each 3 ft. only in length and 3 in. in internal diameter. These tubes are packed throughout with fragments of a suitable refractory material, meshed to the proper size. Into the front end of the tube, where the gaseous mixture is introduced, is fitted a fire-clay plug, through which is bored a circular hole about 0.75 in. in diameter. This plug serves the double purpose of keeping the front end of the boiler cool, and of providing a suitable aperture through which the gaseous mixture may be introduced at a speed much higher than the point of back-firing.

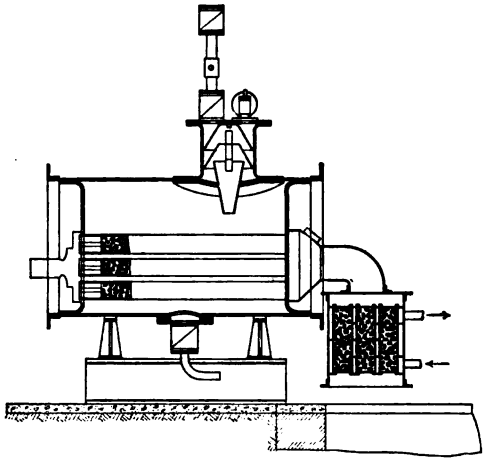

FIG. 3.—EXPERIMENTAL BOILER OPERATED BY FLAMELESS COMBUSTION.

Attached to the front end of the boiler is a mixing-chamber of special design, not shown in detail in the figure. The mixture fed into the boiler-tubes from this chamber consists of the combustible gas, with a proportion of air slightly in excess of that required for complete combustion. The mixture is injected by pressure, or drawn by suction, through the orifice in this fire-clay plug, upon the incandescent material in the tubes. The combustion of the mixture in contact with the incandescent material is complete before it has traversed about 6 in. of the tube from the point of entry. The result is that the core

of the material at this part of the tube is maintained at a high temperature, although the areas of actual contact between the hotter material and the walls of the tube are so rapidly cooled by the transmission of heat to the water in the boiler, that they never attain a temperature even approaching red-heat.

The combustion having been completed, the remainder of the material acts as a baffle towards the burnt gases as they traverse the tubes at a high velocity, causing them to impinge repeatedly on the walls of the tubes. The usual rate at which the gaseous mixture is fed into the boiler corresponds to the hourly consumption of about 100 cu. ft. of coal-gas plus six times its volume of air for every tube of the boiler, or an equivalent volume (*i. e.*, equivalent as regards heating-capacity) of any other gaseous mixture. Thus, for the ten-tube boiler on which the original experiments were made, the consumption of coal-gas was about 1,000 cu. ft. per hour, plus about 5,500 or 6,000 cu. ft. of air. These figures indicate the extremely rapid rate at which the mixture is caused to traverse the tubes.

Utilization of the Heat in the Exit Gases.

After the burnt products have traversed the boiler-tubes, their temperature is never more than about 70° C. above that of the water in the boiler (which, of course, depends upon the pressure at which the steam is being generated). This is a much lower temperature than that at which the products of combustion usually pass away from a multitubular boiler. But, in order to increase still further the output of steam, the products are passed through a short tubular feed-water heater, constructed on the same principle as the boiler. During a test carried out in Leeds, in which steam was generated at 100 lb. above atmospheric pressure, the temperature of the products leaving the boiler-tubes was 230° C.—the actual boiling-point of the water being 170° C. These products, still containing a certain amount of valuable heat, were passed through a feed-water heater, only 1 ft. long, containing nine tubes, of the same diameter as those in the boiler, packed with granular material. The hot products, continually baffled in their passage through the tubes, readily imparted their heat to the cold feed-water surrounding them; and their temperature was thereby reduced to somewhat less than 100° C.

The Ten-Tube Experimental Boiler.

The connections to the front of this boiler consisted essentially of a tube for the supply of gas, and another for the supply of air. The gas and the air were mixed before entering the feeding-chamber attached to the front plate of the boiler; the gaseous mixture was burned in the tubes of the boiler; and the products passed outward at the other end into a small chamber, and thence into the feed-water heater.

The mixture of gas and air was passed into the feed-chamber of this boiler at a pressure of 17.3 in. water-gauge. This pressure was necessary in order to overcome the resistance of the packing of the tubes. The pressure of the products entering the tubes of the feed-water heater was 2 in. water-gauge, so that the pressure necessary to force the gas through the zone of combustion, and thereafter through the remainder of the boiler-tubes, was about 15 in. water-gauge. In carrying out the test the water was evaporated at 100 lb. above atmospheric pressure; the temperature of the boiling water was therefore 168° C. (or 337° F.). The temperature of the combustion-products leaving the boiler-tubes was 230° C. The average temperature of the products leaving the feed-water heater was 95° C., or 203° F. The temperature of the water entering the feed-water heater was 5.5° C. (or 42° F.), and it was heated to 58° C. (or 136.4° F.) before entering the boiler, entirely at the expense of the burnt gases.

The ratio between the heat transmitted to the water and the net heat of combustion of the burnt gas in the boiler was 0.94; *i. e.*, more than 90 per cent. of the heat generated was utilized.

It is one of the prominent merits of the new system that the gas is burned completely with a minimum excess of free oxygen. During the test in question, the average proportion of carbon dioxide in the combustion-products was as much as 10.6 per cent., while the oxygen was as low as 1.6 per cent. The most careful examination of the products failed to reveal the presence of the slightest trace of carbon monoxide, hydrogen, or methane. Therefore, the remainder of the gas was simply nitrogen. Even with as little as 0.5 per cent. of oxygen in the products, the combustion of the gas in the tubes is perfect, not a trace of combustible gas escaping.

The Skinningrove 110-*Tube Boiler for Coke-Oven Gas.*

That the gas-firing of boilers according to the new system has been advanced beyond the merely experimental stage is proved by the recent erection by the Skinningrove Iron Works Co., Ltd., Chuland, Yorkshire, of a 110-tube boiler capable of evaporating not less than 5,500 lb. of water per hour, fired by gas from a new installation of coke-ovens adjacent to the blast-furnace. As shown in Fig. 4, this boiler is a cylindrical drum 10 ft. in diameter, and only 4 ft. from front to back; it is traversed by 110 tubes of 3 in. internal diameter, packed with fragments of fire-brick. It is worked under the suction

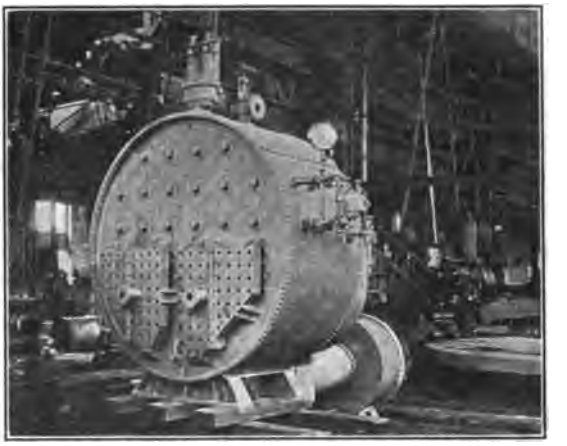

FIG. 4.—SKINNINGROVE GAS-FIRED BOILER OPERATED BY FLAMELESS COMBUSTION.

of a fan. To the front is attached a device whereby gas at 2 in. water-gauge pressure from a suitable feeding-chamber, together with a proper proportion of air from the outside atmosphere, is drawn (under the suction of the fan) through a short "mixing-tube" into each of the 110 tubes of the boiler, where it is burned without flame, in contact with the incandescent granular material. The products of combustion, having traversed the 4 ft. of packed tube, pass outward into a semicircular chamber at the back of the boiler, and thence through a duct to the tubular feed-water heater, represented in Fig. 5. A fan attached to the feed-water heater removes the cooled

products and discharges them through a short duct into the atmosphere outside the boiler-house.

In construction, nothing could be simpler or more compact than a cylindrical shell only 4 ft. long by 10 ft. in diameter, supported on a casting and requiring neither elaborate brick-work setting nor chimney. The boiler has the further structural advantage over all other multitubular boilers, that the front plate can never be heated beyond the temperature of the water, however much the firing may be forced. This circumstance, coupled with the extremely short length of the tubes, implies an absence of strain and greatly reduces the risk of

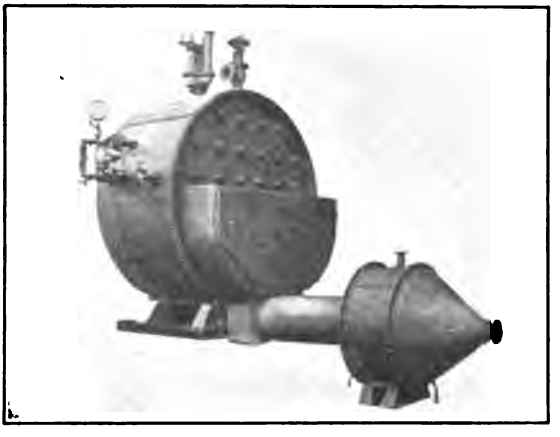

FIG. 5.—FEED-WATER HEATER OF THE SKINNINGROVE BOILER.

leaky joints. Another feature of the boiler which makes for efficiency is the steep "evaporation gradient" along the tubes. Under the normal working-conditions, the "mean evaporation" exceeds 20 lb. per sq. ft. of heating-surface, or about twice that of a locomotive boiler. Of the total evaporation, no less than 70 per cent. occurs over the first third of the tubes, 22 per cent. over the next third, and about 8 per cent. over the remainder. Such a steep gradient causes a considerable natural circulation of the water in the boiler, a factor of great importance in good working. As to thermal efficiency, it seems reasonable to expect that a boiler-unit which, while evaporating 20 lb. of water per sq. ft. of heating-surface, transmits upward of 90 per cent. of the net heat of combustion of

the gas to the water, and which, if need be, can be forced to a 50 per cent. higher "duty," with only a slight drop in efficiency, will stand unrivaled as a steam-raiser. Moreover, in the case of a large boiler of (say) 100 tubes, "elasticity" may be conferred by arranging the tubes in groups, so that they may be fired up or completely shut off, group by group, successively, in correspondence with variations in the load.

The Skinningrove boiler has proved almost completely automatic in its working, according to the statement of the manager of this plant, who says also: "The boiler has been off for the inspection of the tubes, which prove to be clean and free from scale, a fact which I attribute to highly rapid ebullition. During the length of time the boiler has been at work, we have had no trouble with priming; the steam having been at all times perfectly dry. The average temperature of the waste gases leaving the plant has been from 78° to 80° C., which is ample proof of the boiler's efficiency."

Experiments to determine the value of this type of steam-generator as a waste-heat boiler show important economies. Work on oil-fired boilers has also been carried out with satisfactory results.

V. The Melting of Easily Fusible Metals and Alloys.

It will be readily understood that the principle embodied in the boiler is capable of great extension. Thus, for example, it can be applied to (1) the preliminary concentration of dilute solutions and the heating of liquids generally; (2) the heating of large volumes of air; and (3) the melting of easily fusible metals and alloys.

I will here refer briefly to some experiments on the fusion of metals. Professor Bone's attention was first drawn to this subject by experts of one of the London gas companies, who represented that there would be a large field of usefulness for the process in melting type-metal for large newspapers, which require for their machines a continuous supply of molten type-metal. Fig. 6 represents an iron tank, efficiently lagged and filled to the top with molten lead at a temperature of, say, 50° above its melting-point. In the molten bath is fixed an iron tube, 2 or 3 ft. long and 3 in. in internal diameter. The tube is packed (like one of the boiler-tubes) with a suitable granular

refractory material, and there are suitable arrangements for the introduction of the explosive mixture of gas and air which is to be burned in the tube. When once the device is started up, it can be worked continuously for days together. Solid lead is continuously fed into the apparatus, and the molten metal is allowed to run over through the spout indicated in the diagram. Experiments have been carried out with tanks holding up to 8 tons or more of molten metal, in which a series of combustion-tubes are fixed. By means of such an apparatus, lead (or other fusible metals or alloys) may be melted not only very rapidly, but with extraordinary efficiency. The following is the result of a test carried out at the experimental station, with a single-tube apparatus:

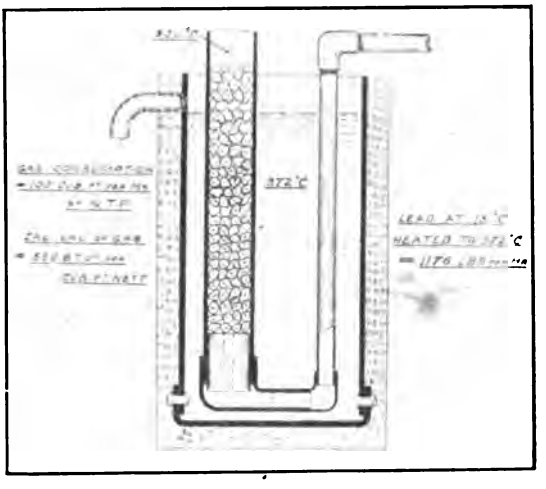

FIG. 6.—TANK FOR MELTING METAL, HEATED BY FLAMELESS COMBUSTION.

Lead-Melting Test.

	Degrees C.	Degrees F.
Temperature of metal charged,	15	60
Temperature of metal tapped,	372	682
Temperature of gases leaving apparatus,	500	932

Lead melted per hour = 1,176 lb.

Heat required per hour to raise metal from 15° C. to 372° C. = 1,176 × 32.67 = 38,420 B.t.u.

Gas burnt per hour = 100 cu. ft. at N.T.P.

Net calorific value of gas = 559 B.t.u. per cu. ft. at normal temperature and pressure.

$$\text{Ratio} = \frac{38.420}{55.900} = 0.686.$$

The conditions were so arranged that the mean temperature of the molten metal in the apparatus was 372° C. throughout the test. Lead ingots, each weighing about 30 lb., were added at intervals of 1.5 min., and the molten metal displaced was simultaneously run off into molds. Great care was taken to keep the bath thoroughly molten, and at a temperature within a few degrees of the mean value. Burning gas of net calorific value of 559 B.t.u. per cu. ft., at the rate of 100 cu. ft. per hour, it was found possible to raise the temperature of 1,176 lb. of lead per hour from 15° to 372° C.; the temperature of the products of combustion leaving the tube being constant at 500° C., or only 128° above the temperature of the molten metal. Using the latest determination by Spring of the specific heat of lead at temperatures up to and above its melting-point, and adopting the usually accepted value for the latent heat of fusion of lead, Professor Bone estimates that at least 70 per cent. of the heat developed was utilized. My observations of this type of heating-apparatus, in comparison with externally heated melting-pots, show a great difference in fuel-consumption in favor of internal heating.

Many other applications of flameless combustion are undergoing exhaustive investigation by Professor Bone and his able colleague, C. D. McCourt; and in the near future we may look for further interesting developments within the many departments of the field of combustion without flame.

Electric Hoist at Hecla Mine, Burke, Idaho.

BY E. M. MURPHY, BURKE, IDAHO.

(Presented by invitation at a meeting of the Spokane Local Section of the Institute, February, 1912.)

EIGHT years ago the Hecla mine, a lead-silver producer, situated at Burke, Idaho, was producing ore from but two levels, the 300- and the 600-ft. All hoisting was done with a 12- by 16-in. slide-valve, double-reel, hoist. The hoist was being worked to its capacity and was unsuitable for hoisting from greater depths. As the work of opening a new level was about to be started, it became necessary to remodel the then-existing equipment or install a new hoist. The recent advent of electricity from Spokane had made available a cheaper power than steam. By a series of tests conducted throughout the Cœur d'Alène district, it was found that it cost $109 per year to generate a horse-power with steam. Electric power was to be had for $50 a horse-power per year. It was decided to substitute for the engines a motor-drive, of sufficient capacity to operate from the 900-ft. level, and ultimately install an entirely new hoist. After disconnecting the engines, installing a new shaft, and making a few minor changes, a 300-h-p., 2,300-volt, 514-rev. per min., variable-speed motor of the induction type was geared to the reel-shaft. The reels were supplied with ⅝- by 4-in. flat rope, and the duty of the hoist was about as follows:

Cage, 1,700; ore, 2,600; car, 1,400; rope, 2,250; total, 7,950 pounds.

The Bristol ammeter chart, Fig. 1, shows the combined current-consumption of the hoist- and mine-motors. The large intermittent load is clearly shown; this produced harmful fluctuations in the voltage of the long, lightly-loaded, 60,000-volt transmission-line from Spokane to the Cœur d'Alènes. The power company received no compensation for the increased cost of producing power for this intermittent or fluctuating load. In order to protect its system from similar excessive

fluctuations of voltage in the future, the power company established a limit to the capacity of motors which it would permit to operate intermittently on its lines.

When planning for its new hoist, it was therefore necessary for the Hecla Co. to adopt an installation which would store energy during the periods when the demand for power is small, and return it during peak-loads. To meet this condition, it

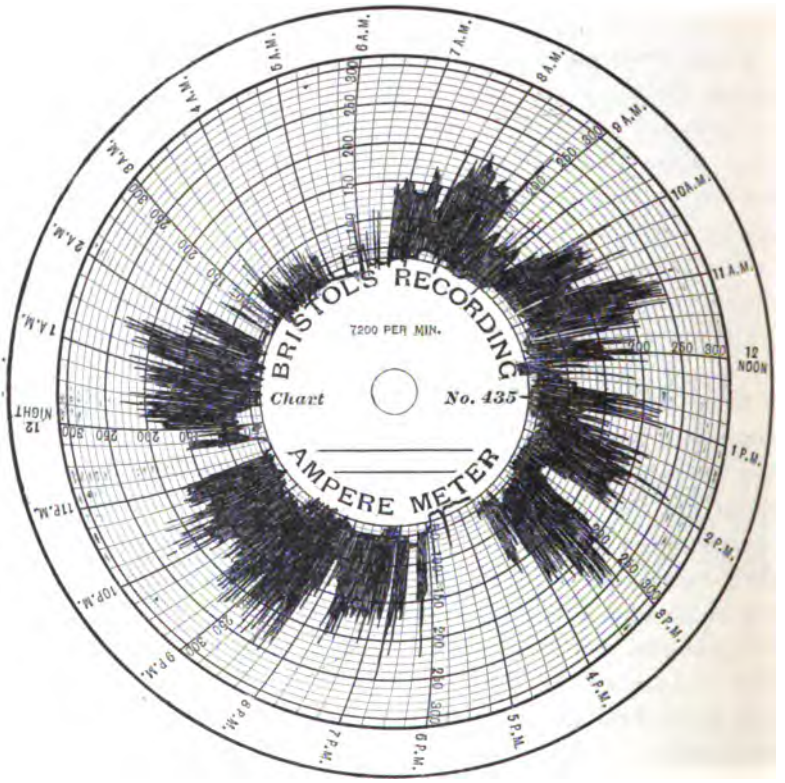

FIG. 1.—CURRENT-CURVE SHOWING LARGE PEAKS PRODUCED BY 300-H-P. INDUCTION-MOTOR OPERATING OLD GEARED HOIST.

was decided to install an electric hoist of the Ilgner type, with the Ward Leonard control.

The new hoist was installed during the spring of 1908, and began actual operations June 1 of that year. The equipment of the motor-generator set is as follows: The set is self-contained, having a cast-iron sub-base, four bearings and shaft; the driving-element consists of a 450-h-p., three-phase, 60-cycle,

wound secondary motor, designed to operate between 2,000 and 2,300 volts.

The generator is a 450-kw., 525-volt machine, with commutation-poles to permit of its handling full-load current at any voltage between zero and maximum. Mounted on the shaft is a fly-wheel, 7 ft. 9 in. in diameter, weighing 29,000 lb. This fly-wheel consists of a cast-steel spider, upon which is assembled a laminated steel rim. Upon the end of the shaft is mounted a 25-kw., 125-volt exciter, the armature of which is on the shaft of the set. The total fly-wheel effect of the set is 220,000 lb. at 1 ft. radius at 720 rev. per minute.

The hoist-motor is rated at 375 h-p., 500 volts, 60 rev. per min., and weighs 51 tons. It is supplied with two bearings and is coupled to the reel-shaft through a forged flanged coupling. The reels are 5 ft., smallest, and 12 ft. 8 in., outside diameter. Each reel is fitted with a Lane clutch and a post brake. At present each reel carries 1,600 ft. of $\frac{3}{8}$- by 4.5-in. flat rope, this being sufficient to operate from the 1,200-ft. level. The center of the reels is placed 93 ft. 10 in. from the center of the shaft. Lack of available space, preventing the placing of the hoist further away, led to the adoption of flat rope. The sheave-wheels are 8 ft. in diameter, and the centers are 102 ft. from the collar of the shaft. The dump is 55 ft. 6 in. from the collar, and the skip-pockets are 25 ft. below the station at the 600- and 900-ft. levels, and 30 ft. below at the 1,200-ft. level. The skips are of 50 cu. ft. capacity, weigh 3,500 lb., and carry 2.6 tons of ore per trip. Double-deck cages hang beneath the skips at all times; each cage weighs 2,400 lb. The motor-generator set, the slip-regulator, and switch-board, are situated in a room adjacent to the hoist-room. On the switch-board are placed the following instruments and equipment: one polyphase indicating wattmeter; one polyphase integrating wattmeter; one A. C. voltmeter; one A. C. ammeter; one double-reading D. C. voltmeter; one D. C. ammeter in the motor field; one D. C. voltmeter on the exciter circuit; one graphic recording D. C. wattmeter; one Tirril regulator for exciter voltage; one DPST switch in the armature circuit of the exciter; one automatic, hand-operated, three-phase, 2,300-volt, oil circuit-breaker; one generator-field switch; one exciter-field rheostat; one generator-field rheostat; one motor-field rheostat; one switch in

the magnet circuit of the slip-regulator switches; two line-relays for the slip-regulator; one compensator for line-relays; current and potential transformers for the various instruments.

Principle of Operation.

In order that the fly-wheel should give up part of its energy during the peaks, it is necessary to have some method of slowing down the set as the load comes on. This is accomplished by what is known as the slip-regulator. This device consists of a sectionalized resistance, comprising 13 sections, connected to the secondary of the induction-motor. Each section can be short-circuited by means of an oil-switch operated by compressed air. The air-valves are controlled by A. C. magnets wound for 50 volts, the magnets being supplied with power from a 2,300- to 50-volt transformer connected between the motor and its circuit-breaker. The magnet circuit comprises three circuits, viz., accelerating, drop-out, and hold-in; each of the first two has a switch in series, which is opened and closed by its respective relay. The relays receive power from a compensator which is excited from two current-transformers in the A. C. motor circuit. The relay switches can be set to operate at any desired value, within certain limits, by means of a dial-switch on the compensator, the one on the accelerating circuit being set to operate on a lower current than the one in the drop-out circuit.

With the A. C. circuit-breaker open, all the short-circuiting oil-switches are out, and the total resistance is in the secondary. To start the set, all that is necessary is to close the circuit-breaker; the starting current, being above that for which the relays are set, operates them, thus opening the accelerating and drop-out circuits. All the oil-switches, of course, remain open. As the set speeds up, the current diminishes and the relays close, causing an oil-switch to be pulled in. This short-circuits a section of resistance, the current increases, and the accelerating relay opens. As long as it remains open, the oil-switches remain out, with the exception, of course, of the one just closed, and this one remains closed, due to the action of the hold-in circuit. When the accelerating relay closes again, due to the fall in current, another oil-switch is closed. The opening and closing of the accelerating circuit, due to the increase and de-

crease of current, with its attendant closing of oil-switches, goes on until all the oil-switches are closed, and all of the resistance is cut out of the secondary. In hoisting, the reverse of the above goes on. As the load comes on, the relays open, and the last oil-switch to come in is the first to go out. This introduces resistance into the secondary, and the motor slows down, letting the fly-wheel take some of the load. Should the current rise again above the set value, another switch is dropped out; this operation can continue until nine switches drop out; four of the 13 are not connected to the drop-out circuit, and are used in starting only.

The hoist-motor is shunt-wound and is separately excited from the exciter. The current in the field can be adjusted by means of the rheostat in its circuit; when the proper adjustment is obtained, it is not necessary to change the setting. The motor is now operated with a field current of 90 amperes.

The Tirril regulator is seldom used to keep the exciter voltage constant, as the drop in voltage during balanced hoisting is found not to be objectionable. In unbalanced hoisting its use is essential, but the wear on the contacts is severe.

The control of the hoist-motor is effected through the field of the generator. The generator's armature is connected direct to that of the motor; any change or reversal in the generator voltage is accompanied by a similar change or reversal in that of the motor. The control resistance is situated under the hoistman's platform, and has a large number of points divided into four sections, connected up after the manner of the well-known Wheatstone bridge. In the off position of the controller a perfect balance exists, and no current flows in the field of the generator; the movement of the controller disturbs the balance, and this diverts voltage from the resistance circuit to the generator field. The controller is operated with a lever and quadrant similar to those on the brakes and clutches; moving the lever away from the operator starts the reels turning away from him, while a movement of the lever towards him starts the reels in that direction. In the generator-field circuit, and located back of the switchboard, is a rheostat for adjustment purposes; when once set, this needs no further attention. Mounted in front of the operator is a double-reading D. C. ammeter, which enables him to see the load on the hoist-motor at all times.

A diagram of connections as the equipment was originally installed is shown in Fig. 2, which does not, however, show the Tirril regulator and the D. C. graphic recording wattmeter. The exciter-field switch also is not shown, since it was installed after the hoist was in operation.

As originally equipped, the hoist had several devices, since discarded, which were intended to add to the safety of its operation and prevent abuse of the equipment by carelessness. A careless operator should not be tolerated on a hoist. The best safety-appliance on any machine is a careful manipulator. In connection with the controller was an automatic slow-down, designed to slow the hoist down as it approached the dump. Geared to the reels were two limit-switches, which controlled the opening and closing of a contactor. This contactor normally short-circuited a resistance in series with the generator field; within a certain predetermined distance of the dump, this contactor opened and the resistance was introduced into the generator field. The cutting in and out of this resistance was accomplished so suddenly that the commutation of the generator was affected. In addition, the failure of the apparatus to work at any time might have caused a serious accident, and it was deemed advisable to discard it.

A circuit-breaker was installed in the armature circuit of the exciter. Connected to this breaker, so as to open it, was an over-speed device on the motor-generator set, it being assumed that during unbalanced lowering excessive speed might be obtained, due to the pump-back action. In practice, this high speed is not developed. An over-load trip and an under-load trip, in the motor-field circuit, could also open the breaker. There are times when the failure of the D. C. power might prove disastrous and be attended with fatalities. It is essential to prevent any accidental opening of the circuit by using nothing but manually-operated switches. The breaker was removed and a switch installed. The over-speed device was connected so as to light a lamp, in front of the hoistman, in case the set speeded up. Should the occasion arise to cut the D. C. voltage off suddenly, it can be done by opening an exciter-field switch, installed in front and within easy reach of the hoistman. In Fig. 2, *X*, *Y*, and *Z* are the parts removed.

The hoist is also equipped with an air-operated device for

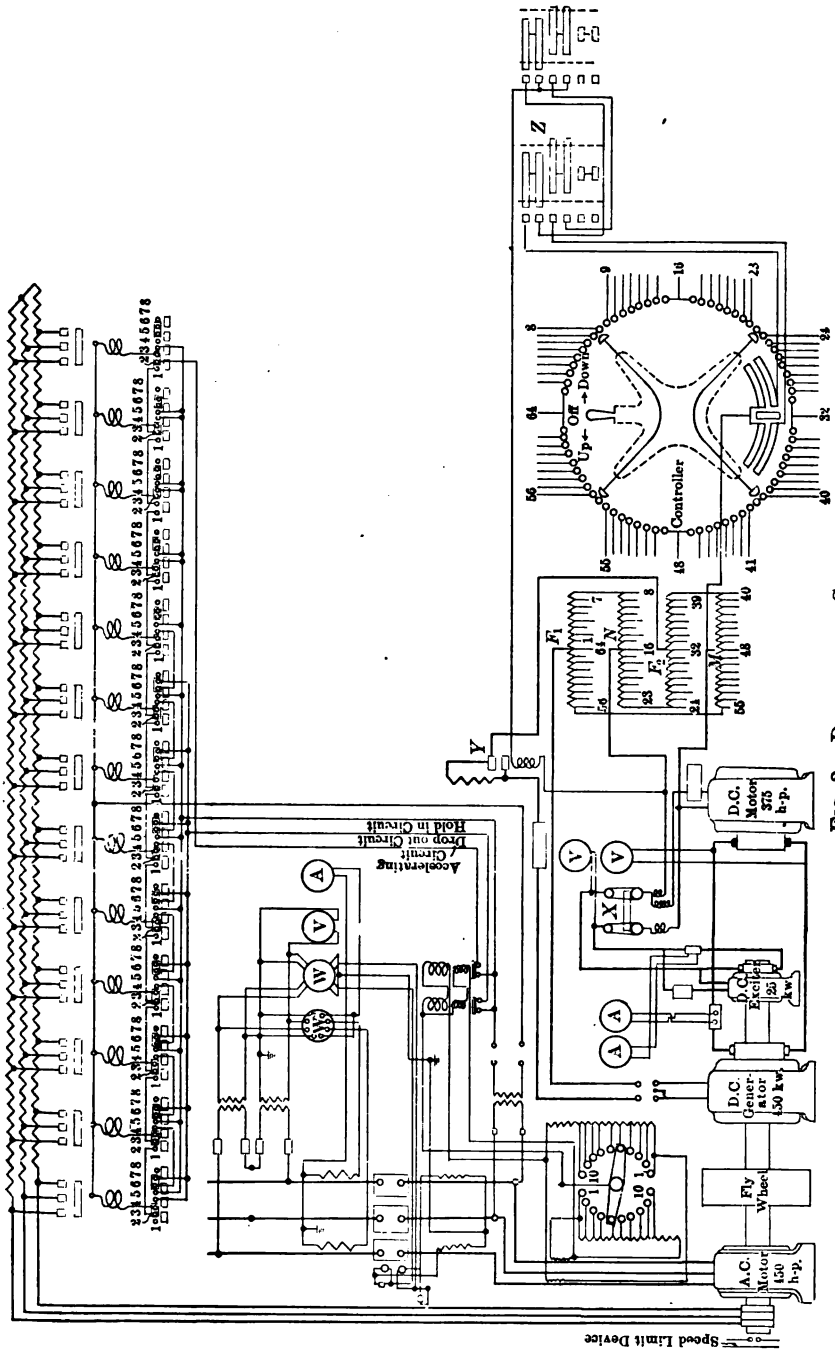

FIG. 2.—DIAGRAM OF CONNECTIONS.

throwing out the clutches and applying the brakes in case of an over-wind. Owing to the lack of sufficient head-room, this apparatus cannot be used. As a substitute, converging or "tight" guides are used, in addition to which are spring hooks, which engage the crosshead and prevent the falling of the skip, should the cable break after the skip reaches the sheaves.

Cost of Operation.

Before entering into the cost of operation of the hoist, an explanation of the present contract, signed in 1908, will serve to show on what basis a settlement is made for power consumed by it. The contract runs for a period of five years and is based on a maximum demand as well as a kw-hr. consumption. It will be noted that it penalizes a better combined power- and load-factor than 61 per cent. A power-factor of 100 per cent. and a constant voltage of 2,300 volts is assumed in all the calculations of maximum demand. In a contract calling for a maximum demand of 100 kw., a minimum sum of $335 ($3.35 per kw.) is paid each month; this sum entitles the consumer to 43,550 kw-hr. (130 for each dollar). At the same time, the maximum of 100 kw. must not be exceeded at any time during the month. In case more than 100 kw. is the maximum, the minimum bill is increased by $3.35 for each kw. in excess. For each dollar of this increase, the consumer is entitled to use 130 kw-hr. If the kw-hr. used exceed 43,550, the excess is paid for at the rate of $0.0112 per kw-hr. On the basis of 43,550 kw-hr. per month at $335 each kw-hr. costs $0.00769. This amounts to $50 per h-p. per year. The peak on the hoist never lasts 5 min., so the power never costs more than $50 per horse-power.

As the Hecla has but one hoist, the handling of all timbers, waste, etc., as well as the shifts, must be performed by it, in addition to the ore-hoisting. To give an idea of the work the hoist does, the following table was compiled for the period of time from Aug. 1, 1911, to Jan. 1, 1912:

Ore Hoisted.

	600-Ft. Level.	900-Ft. Level.	1,200-Ft. Level.
Skips,	2,241.0	9,968.0	9,961.0
Tons,	5,840.0	25,824.0	26,028.0
Monthly Average.			
Skips,	448.2	1,993.6	1,992.2
Tons,	1,168.0	5,164.8	5,202.6

Cars of Waste Handled.

	1,200 to 600.	Top to 600.	Top to 300.
Cars, ...	3,946.0	7,416.0	826.0
Average, ...	789.2	1,483.2	165.2

	Stulls.	Wedges.	Timbers Lagging and Chute.
	321,677 ft. B.M.	79 cars	475,140 ft. B.M.
Averages,	64,335.4	15.8	95,028.0

Power Consumed: 234,760 kw-hr., average equals 46,952 kw-hr., equals $361 per month. Total cost for power for each ton of output equals $0.0313.

In order to determine the cost per ton for power used during actual hoisting, a series of tests was taken, with the following results: From the 1,200-ft. level 32 skips (83 tons) were hoisted in 33 min., with a kw-hr. consumption of 142. On the basis of $0.00769 per kw-hr., the cost of hoisting the 83 tons was $1.092, or $0.0131 per ton. From the 900-ft. level 14 skips (36.4 tons) were hoisted in 11 min., with a kw-hr. consumption of 50, and a cost of $0.3854, or $0.0105 per ton. From the 600-ft. level 14 skips (36.4 tons) were hoisted in 10 min., with a kw-hr. consumption of 40, and a cost of $0.3076, or $0.00845 per ton.

To run the set light for 1 hr. requires 48 kw-hr., at a cost of $0.368. In service the set runs continuously during the 24 hr., with the exception of a period of about 4 hr. after midnight. After the power is cut off, the set will run for 1.25 hr., unless it is slowed down by hoisting, or the band-brake on the fly-wheel is applied. The hoist was guaranteed to maintain one-quarter output of the mine working unbalanced from the 2,400-ft. level (its ultimate depth). In order to test this feature, a load of 1,773 lb. was added to compensate for the extra weight of cable to 2,400 ft. This weight was obtained by placing a car with the required amount of ore in it on a cage-deck. The car was allowed to remain on the cage during the entire time of hoisting. Unbalanced hoisting was maintained at the rate of 11 trips an hr. from 900 ft. for 3 hr. All temperatures at the end of this time were well within the guarantees. Figs. 3 and 4 show 5-sec. readings for one complete trip during this test. In May, 1911, one of the clutch-arms broke, and the hoist operated unbalanced with entire satisfaction for a period of 20 hr., part of the hoisting being from the 1,200-ft. level.

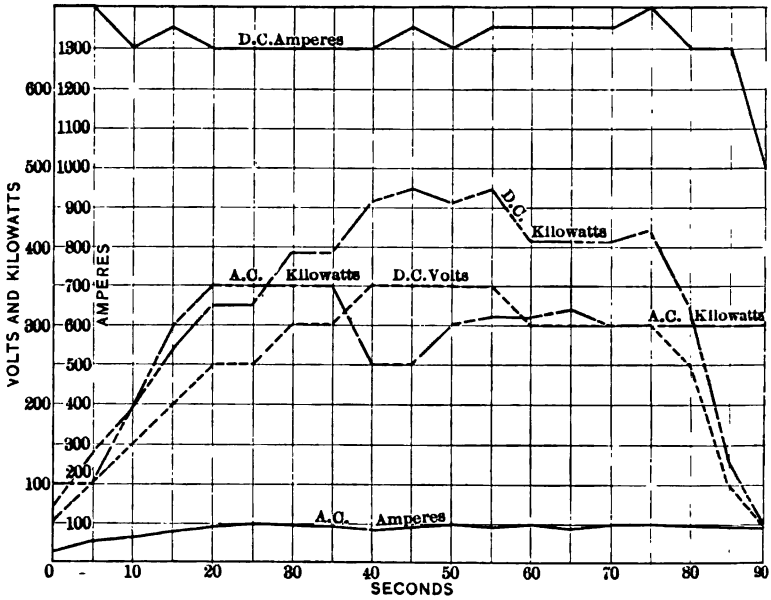

FIG. 3.—FIVE-SECOND READINGS TAKEN DURING UNBALANCED HOISTING TRIP FROM 900 LEVEL. SUFFICIENT EXTRA WEIGHT ADDED TO APPROXIMATE HOISTING-CONDITIONS FROM 2,400 LEVEL. RESISTANCE-SWITCHES OPERATE AT 310 KW. ALL RESISTANCE IN AT END OF TRIP.

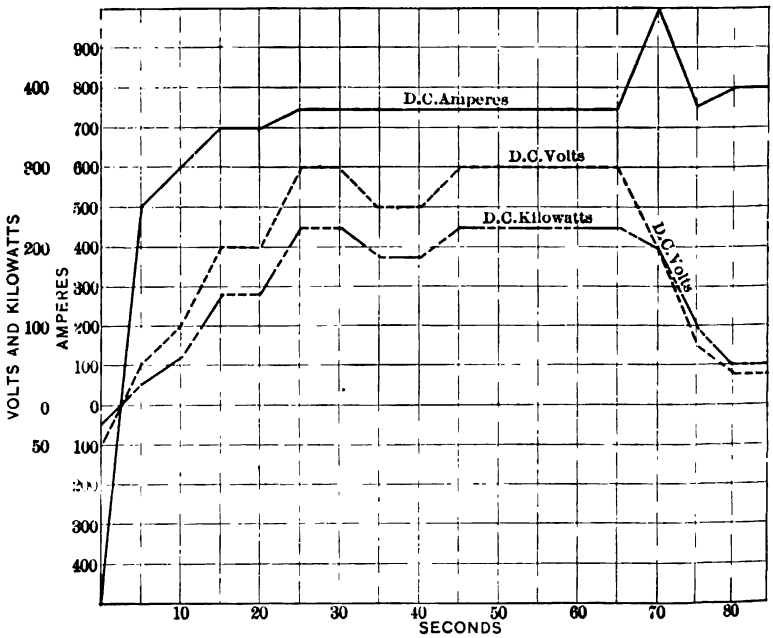

FIG. 4.—FIVE-SECOND READINGS TAKEN DURING UNBALANCED LOWERING, WITH EXTRA WEIGHT, TO 900 LEVEL.

Figs. 5, 6, and 7 show 5-sec. readings for unbalanced hoisting from 1,200 and 900 ft. with ordinary load of 5,200 lb. of ore. Figs. 8, 9, 10, and 11 show 5-sec. readings of balanced hoisting from 1,200 ft. and 900 ft. with two different switch-settings. The effect of the different settings is clearly shown in the A. C. kw. input. It would seem from this that it would be desirable to keep the setting that would give the minimum input at all times. The objection to doing this is that the lower

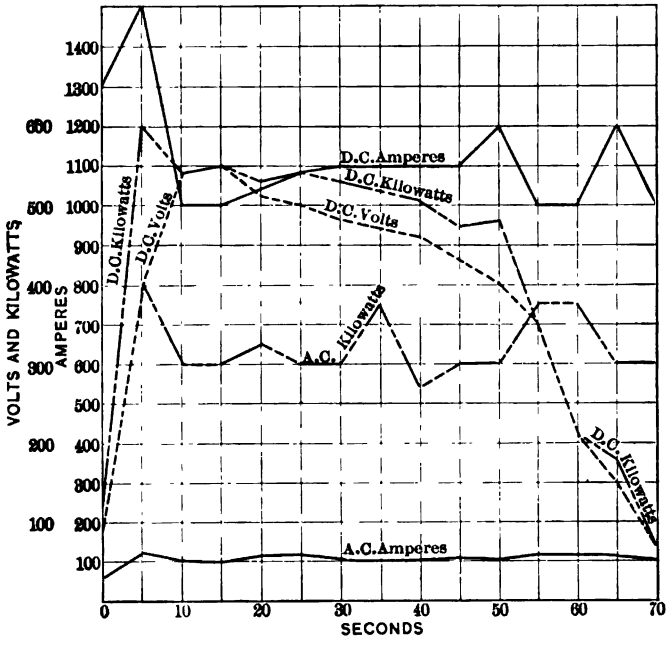

FIG. 5.—FIVE-SECOND READINGS FOR UNBALANCED HOISTING FROM 1,200 LEVEL UNDER NORMAL CONDITIONS. RESISTANCE-SWITCHES OPERATE AT 310 KW. ALL RESISTANCE IN AT END OF TRIP.

the setting, the greater the number of switches that drop out, and the slower the speed of the set becomes. When all nine switches drop out, it takes 55 sec. for them to come in again and for the set to accelerate to full speed. It takes from 6 to 8 sec. to load and dump a skip, and if the speed in hoisting is not to be sacrificed, the switch-setting should be such that the number that drop out can come in again in the time it takes to load and dump. In practice this is found to be about three.

The setting, of course, has to be made for the lowest level hoisted from, or else the setting be changed for different levels.

The upkeep of the equipment for the three years and eight months it has been in service has been extremely low. The hoist-motor has needed no repairs, while the exciter has had but one new set of brushes. The generator requires about one new set of brushes a year. The A. C. motor has been the only

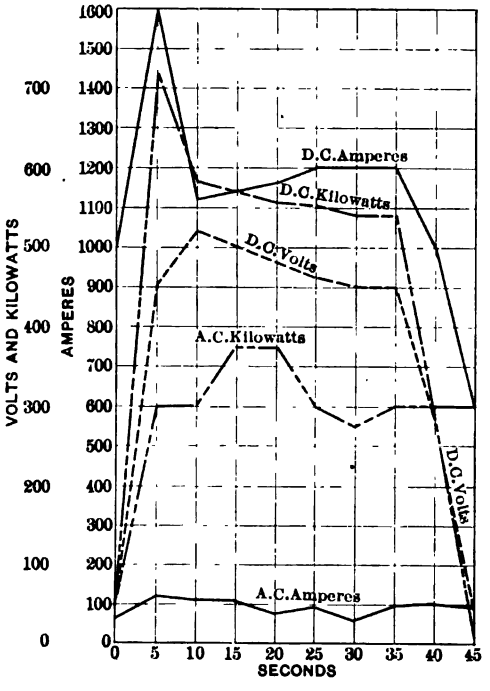

FIG. 6.—FIVE-SECOND READINGS FOR UNBALANCED HOISTING FROM 900 LEVEL UNDER NORMAL CONDITIONS. RESISTANCE-SWITCHES OPERATE AT 310 KW. ALL RESISTANCE IN AT END OF TRIP.

source of expense, and like trouble could occur to any motor. Three times it has suffered a grounded coil during a lightning storm. The winding is a three-bank concentric winding, and replacement of coils is a tedious affair. A new set of collector-rings was also put on this machine last December.

One noticeable feature of this installation is its noiseless operation and the facility of its control. The motor can be reversed in an incredibly short space of time, and during load-

ing and dumping the brakes are seldom applied. The wear of the brake-blocks is inappreciable, and the original set will no doubt last the life of the mine. The effect of the smoothness of operation is particularly noticeable on a cable, a service of from 10 to 14 months being obtained from each sewing, while the strands will last for years.

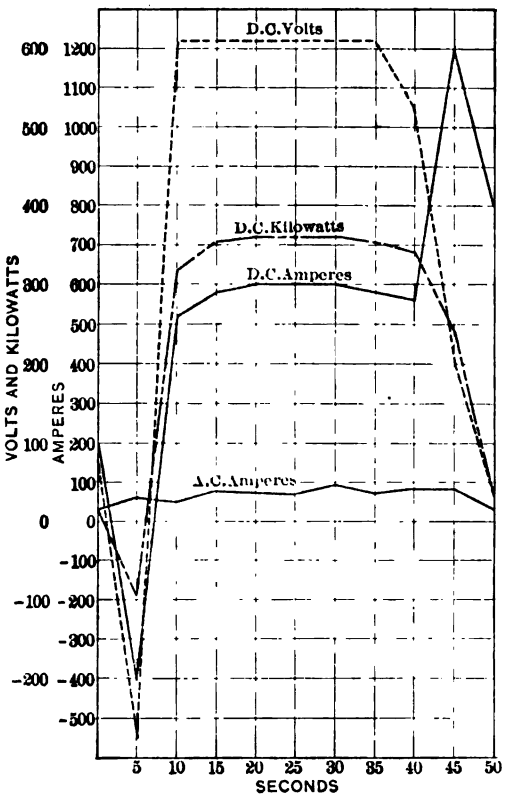

FIG. 7.—FIVE-SECOND READINGS FOR UNBALANCED LOWERING TO 1,200 LEVEL UNDER NORMAL CONDITIONS.

The control is so accurate that no chairs are needed in the shaft to run on and off the waste-cars.

Should the brake be removed when the skip is empty, the speed with which it will descend unbalanced is remarkably low. This feature is clearly shown in Fig. 12, which is a set of readings taken at 1-min. intervals for a trip from the top to

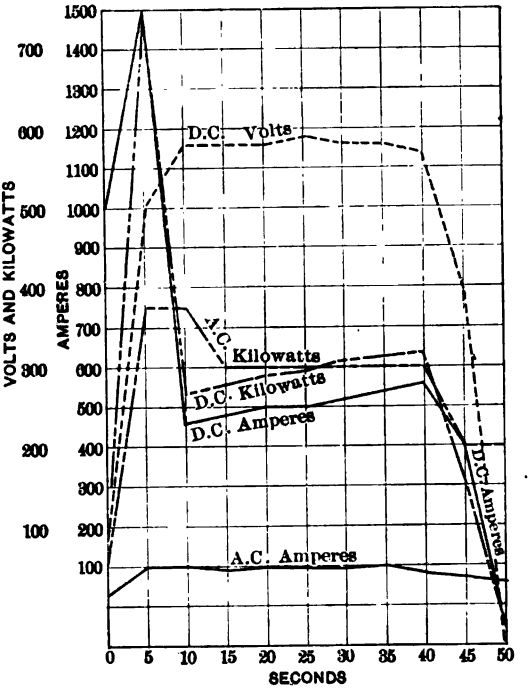

FIG. 8.—FIVE-SECOND READINGS TAKEN DURING BALANCED HOISTING TRIP FROM 1,200 LEVEL. RESISTANCE-SWITCHES SET TO OPERATE AT 310 KW. ONE-THIRD RESISTANCE IN AT END OF TRIP.

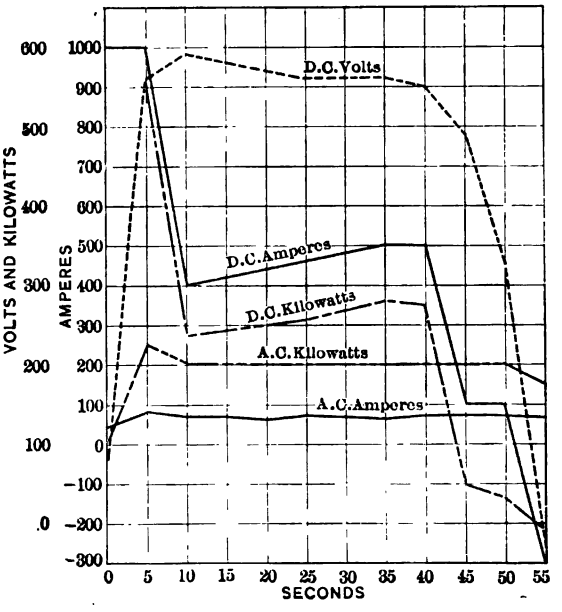

FIG. 9.—FIVE-SECOND READINGS TAKEN DURING BALANCED HOISTING FROM 1,200 LEVEL. RESISTANCE-SWITCHES SET TO OPERATE AT 210 KW. TWO-THIRDS RESISTANCE IN AT END OF TRIP.

300 ft. No matter how slow the speed is, the pump-back action is taking place, and during the trips that waste and timber are being lowered into the mine, power is actually restored to the line.

The hoist requires but one man per shift to operate it. Another advantage of the hoist is its ability to operate for a

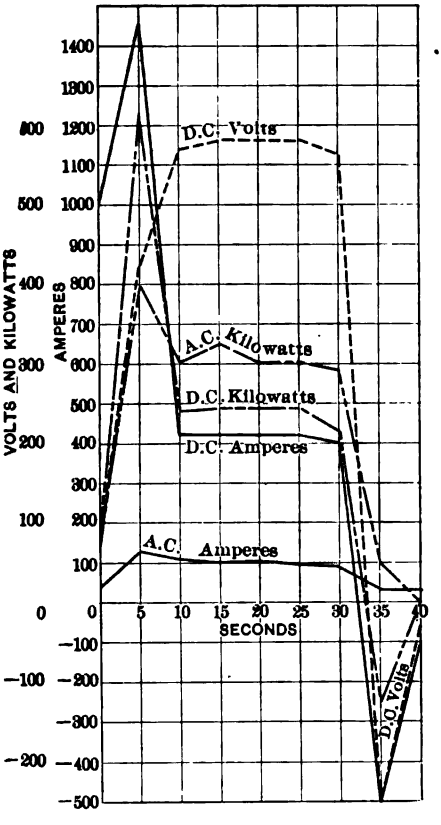

FIG. 10.—FIVE-SECOND READINGS TAKEN DURING BALANCED HOISTING TRIP FROM 900 LEVEL. RESISTANCE-SWITCHES SET TO OPERATE AT 310 KW. ONE-THIRD RESISTANCE IN AT END OF TRIP.

short time, even though the power be accidentally interrupted. The running-lights in the hoist-room are all lighted from the exciter, which enables the operator to see as long as hoisting can be continued. Figs. 13 and 14 show 5-sec. readings for trips from the 900- and 1,200-ft. levels with a skip of ore after the A. C. power is cut off. During these tests, the speed of the

set became so low that the Tirril regulator short-circuited all the exciter-field rheostats, and its magnets failed to work. Without ore, as in handling men, the hoist is capable of making several trips after the A. C. power is off.

This installation has the disadvantage of consuming power,

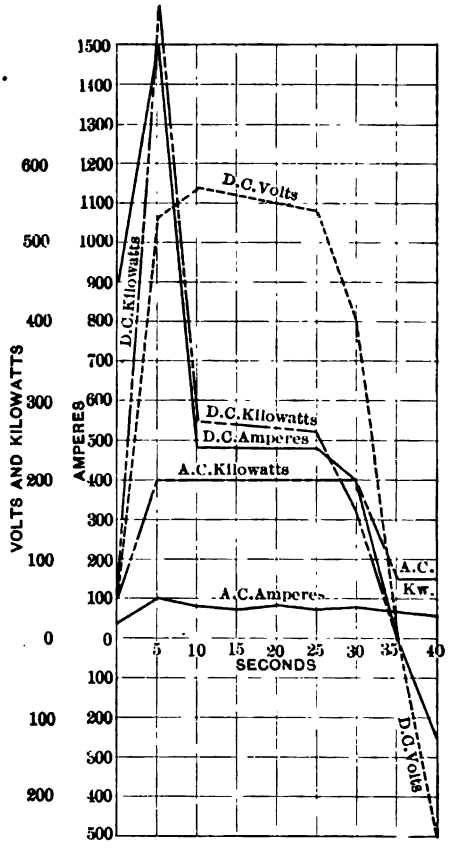

FIG. 11.—FIVE-SECOND READINGS TAKEN DURING BALANCED HOISTING FROM 900 LEVEL. RESISTANCE-SWITCHES SET TO OPERATE AT 210 KW. TWO-THIRDS RESISTANCE IN AT END OF TRIP.

even though the hoist-motor is not in actual operation. This is more apparent than it would be if the hoist were operating from greater depths, or handling greater tonnage. The effect greater depth has on the efficiency is shown from the tests. From the 600-ft. level, the cost per 1,000 ft.-tons is

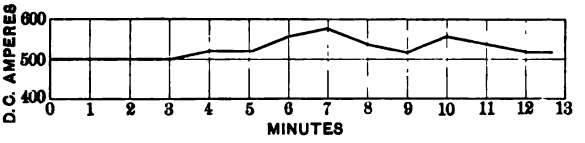

FIG. 12.—ONE-MINUTE READINGS TAKEN FOR UNBALANCED TRIP FROM TOP TO 300 LEVEL. THIS TRIP WAS STARTED AND STOPPED WITH THE BRAKE, THE CONTROLLER REMAINING IN THE OFF POSITION.

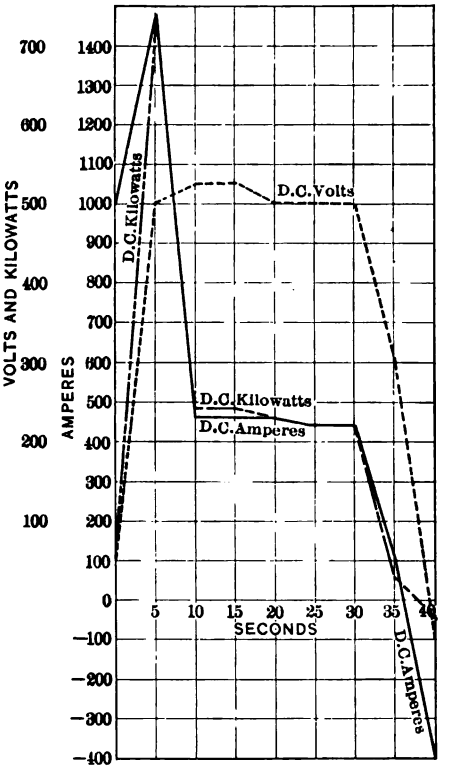

FIG. 13.—FIVE-SECOND READINGS TAKEN DURING BALANCED HOISTING FROM 900 LEVEL WITHOUT A. C. POWER, THE A. C. CIRCUIT-BREAKER BEING OPENED AT THE BEGINNING OF THE TRIP.

$0.014, from 900 ft. it is $0.0116, and from 1,200 ft. it is $0.0109. The effect greater tonnage would have on the cost per ton of output is shown by the following: Assuming that the mine double its output, the kw-hr. consumption per month would be increased by 17,231, at a cost of $132.50. The total

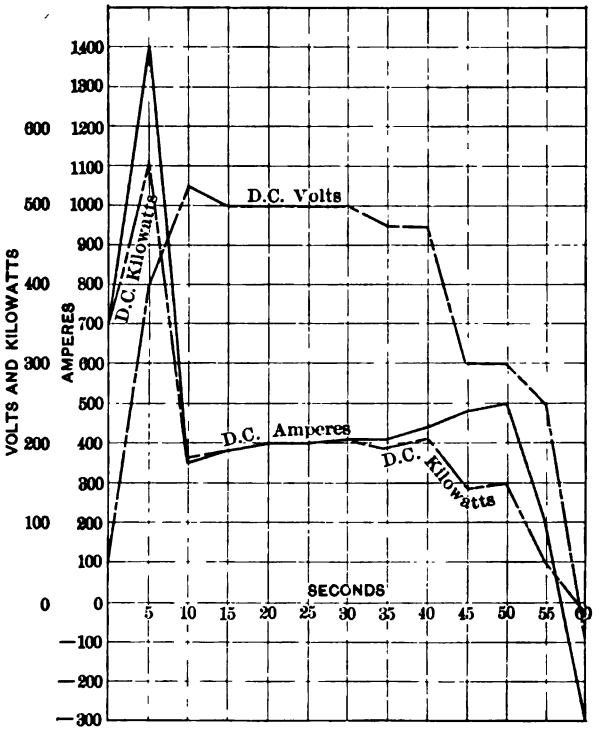

FIG. 14.—FIVE-SECOND READINGS TAKEN DURING BALANCED HOISTING FROM 1,200 LEVEL, WITHOUT A. C. POWER. THE A. C. CIRCUIT-BREAKER WAS OPENED AT THE BEGINNING OF THE TRIP.

cost for power for each ton of output would be lowered from $0.0313 to $0.0213.

Each hoisting problem has its own solution, and the Hecla Mining Co. appears to have solved this one in a most satisfactory manner.

Efficiency-Engineering Applied to Mining.

BY GLENVILLE A. COLLINS, SPOKANE, WASH.

(Presented at a Meeting of the Spokane Local Section of the Institute,, February, 1912.)

BEING much interested in efficiency-work in general, and an ardent reader of writings on this subject by Frederick W. Taylor and others, as applied to manufacturing-plants, and having served my apprenticeship from mucking to the engineering design of ore-dressing and underground plants, I hope that this paper will lead others to investigate the field more fully, to the improvement of general mine- and mill-operations, as well as the advancement, intellectually, physically, and financially, of the employee and his relations with the employer.

If a man who is prepared to look closely and observe broadly starts in Canada and zig-zags from camp to camp to southern Mexico, taking time casually to inspect mines, mills, and smelters, he will be pitifully impressed by the great difference in customs, and the general waste of labor, time, and money throughout the country. He will so seldom observe a dollar's worth of results given in exchange for a dollar spent that he will form a poor opinion of operating-efficiency and management. This highly-scientific department, requiring an amount of skill and executive ability equaled in few, if any, of the professions, certainly falls short of the standard established in our greater manufacturing industries.

It is easily said that this is due to the diversity of conditions in mining, milling, and smelting. But on second view it must be admitted that there is room for tremendous improvements in nearly every mining-organization. The proposal of such improvements is often met by the argument that any mine is but a temporary affair, yielding such profits as to make efficiency-work of no consequence; but any business which is profitable and worth doing at all is worth doing on a basis which gives the greatest return on the investment.

MANAGEMENT.

Neither an extremely dictatorial nor a lax management results in efficient operation in any mine; and harmonious relations between heads of departments are impossible under either of these two conditions. The mining business is such that every one connected with it is apt to be under strain from living-conditions, climate, and extreme duties of many characters, which makes it all the more necessary for the entire organization to be most harmonious.

Any scheme securing harmony of operations and increase of output at less expense to the owners, and still giving the employee better wages and living-conditions, is a benefit to the industry; and the higher development of the industry requires a combination of intelligence and labor, in which low cost of labor is to be desired by the employer and high wages by the workman. Diversity of mining-locations and conditions does not affect this truth.

It is, then, the purpose of this paper to advocate throughout the industry, high wages for workmen and low labor-costs to the employer, or what is generally known to manufacturers as the "Taylor System." I contend that this system is entirely applicable to the mining industry, through careful study and record of the time required to perform certain work.

It is often observed that the manager is a master in some branch of the business, and, as a rule, that branch is very well managed, while at the same time other departments under the same management are sorely neglected.

I would define good management as the accomplishment, expediently and cheaply, of exactly the result desired by the owners. When it is understood that a first-class workman can do from two to four times as much work in a given time as the average workman, the possibility of high wages for workmen with low labor-cost to the employer is appreciated.

It has often been noticed that if a workman is informed that it is possible for him, without undue fatigue, to produce from two to four times his normal work, he is incredulous, to say the least. Yet a really good workman, if given an opportunity to work at greater speed and earn from 20 to 100 per cent. more money, is only too glad to get the chance. But the exact percentage to be gained under this system by such terms of pay-

ment that a workman will do his best, can be determined only from carefully kept records, taken by a trained man, over long periods of time. It is not a matter to be decided off-hand, either by labor-unions or by directors.

The usual first-class workman will prefer to remain at his old system of work and wages rather than work at maximum speed unless the increase of pay is sufficient and permanent. For this reason it is not wise to make sudden or ill-advised changes, as the workmen then lose confidence in the system. It should never be referred to as a trial system, intimating the probability that it might not be a complete success. The thing to do is to be absolutely sure, then do it, regardless of the opinions of workmen. As soon as they begin to see the practicability of increasing their earnings and comforts, their opposition will cease, and they will become highly enthusiastic and thrifty.

As soon as the workman begins to realize that he is dealt with as an individual, with growing opportunity consistent with his energy and intellect, he leaves the ranks of organized shiftlessness and becomes an ambitious worker. This is also of great advantage to the employer, as a process of elimination which often develops extraordinary talents, otherwise locked up within a sullen breast.

The majority of manual laborers naturally make their task as easy as possible, eventually doing only that which they are forced to do. If an energetic workman is put with the rest on gang-work at day-wages, he gradually realizes that he is doing more work for the same pay, and in time slows down to the normal pace of the gang.

This may be exemplified by workmen tramming ore-cars. I have often seen them, on leaving the muck-pile at the face, mount the truck and take advantage of the usual down-grade and quick time to the dump, and then, after dumping the car, sit down on the pile from 5 to 20 min. for a rest, in order to give the muckers time to load easily the next car, and likewise "rest" before the trammers' return.

This form of systematic "soldiering" is often practiced to a remarkable degree undiscovered by managers.

When each operation and its required time for performance is absolutely determined and the workman put upon a just

time-basis of minimum wages for a full shift's-work, with a bonus for extra work within the shift, then soldiering will have been effectually stopped.

As a further example, let us consider the case of drill-operators at work on ordinary drifting, using a machine-drill, but not doing their own mucking. Under the old system the drill-operator draws $4 per shift, and his helper $3. By careful time-study on this kind of work, it is found that, under old methods, a round of about 8 holes is drilled during the shift, with an average depth of 4 ft., making a total of 32 lin. ft. of hole per shift, at a labor-cost of $7, or an average of 22 cents per foot of hole drilled. This work is leisurely performed in the average drift in about 5 hr., after which from 1 to 2 hr. is used to clean out, load and fire the holes, and any balance of time left on the shift is wasted under the excuse of gases and bad air. In fact, there are very few managers who do not consider the above as a very fair day's-work, if they think of it at all.

After careful time-study we find that time is lost in securing proper tools for the job, their adjustment, delays caused by faulty conditions of equipment, and improper or no instructions from the foreman as to the work required, such as the most advantageous way of placing drill-holes. In trying to make it possible for the workman to earn from 20 to 100 per cent. more wages by using advanced methods to avoid loss of time, we find that it is very easy for the same men under the same conditions to put in the 32 lin. ft. of drill-holes, load and blast, in the first half of the shift, using the time required for ventilation in which to eat their lunch and look up any repairs or extras needed for the second half of the shift—the same work to be performed in the second half as in the first.

In the instance given, the driller and his helper are on a basis of $7 per shift and are not allowed to maintain their positions unless an average of 40 lin. ft. of hole is drilled, loaded and blasted per shift; but in excess of this footage a rate of 15 cents per linear foot of hole is paid, provided the excess is sufficient to average the hole-depths for efficient blasting.

This is equivalent to 51 per cent. increase in wages to the workmen and a reduction of 24 per cent. in the labor-cost to the employer; or, to be more explicit, when the crew makes

64 lin. ft. of hole per shift the driller receives $5.80 for his labor and the helper $4.80, while the employer pays $10.60 for labor in place of $14 under the old system.

It is of course necessary, in order to perfect such a system, that the tools be kept in good repair and proper means be provided to expedite mucking, tramming, and other portions of the work, in accordance with the general scheme.

To facilitate office-work in keeping records from which to compile pay-rolls and other data, it has been found economical and advantageous to use a card-system in which the workman makes out his own daily card. A form of card embodying this idea is shown in Fig. 1. The statement of the workman is easily checked by the foreman, who approves it before filing. All cards can be filed daily by dates without being indexed, and only handled once when making up the pay-roll or other work-reports. This file has many other advantages, among which are its accessibility, compactness, and simplicity. Again it serves as a "who's who" index of the best workmen, and requires but one office-man for a reasonably large company, covering mining as well as milling in all phases.

When required, the data on the workmen's cards can be plotted graphically so as to show, by inspection, operating-costs and results, without studying in detail each card. This is of great advantage in making annual reports to directors and stockholders, who usually dislike to study pages of detailed data, and prefer to inspect graphic charts which show at a glance the aggregates, and upon more careful examination the details, all given in a few sheets, instead of many pages of a lengthy and often poorly-written report.

It must be remembered that the average workman does not look far into the future, but demands immediate results for his labor. It is, therefore, very necessary to give workmen a just reward for their labor at the time of its performance, instead of holding out promises for the distant future. Hence it is wise, in establishing this system, to apply it very gradually, but firmly, and with definite data at hand.

On careful inquiry into the condition of workmen now working under this system in manufacturing-plants, it is found that their living-conditions improve greatly. It is almost impossible for one to drink to excess and keep up the work required.

It is in reality a body of only first-class men who are all working together to secure large wages which are theirs irrespective of dull times and labor-strifes. The organization is in reality a labor-union in which the employer pays all expenses and gives the workman the best wages he can earn.

In adapting this system to mining one must expect to spend considerable money in order to secure proper fundamental data and to educate the men to the system. If these matters are given proper consideration, and the work is carried out gradually, step by step, the result will be thoroughly successful.

In mining, where it is desired to make the greatest possible output in the shortest time and at the least expense, it is evident that labor is the most important factor with which we have to deal. I believe that any experienced mine-manager will agree that a strong working-organization is of greatest value to the success of the industry. Mills and machinery which are destroyed by fire can easily be replaced with new capital; but it takes a long time to develop a strong working-organization.

I repeat that, in order to attain the most efficient management it is necessary to combine high wages with low labor-cost. The following four principles, as advocated by Frederick W. Taylor in his paper, Shop Management,[1] seem most simple of attainment:

1. *A Large Daily Task.*—Each man in the establishment, high or low, should have a certain clearly-defined task laid out before him. This task should not in the least degree be vague or indefinite, but should be circumscribed carefully and completely, and should not be easy to accomplish.

2. *Standard Conditions.*—Each man's task should call for a full day's-work, and at the same time the workman should be given such conditions and appliances as will enable him to accomplish his task with certainty.

3. *High Wages for Success.*—He should be sure of large pay when he accomplishes his task.

4. *Loss in Case of Failure.*—When he fails he should be sure that sooner or later he will be the loser by it.

It is most advisable to organize what may be called a "planning"-department. I would advocate that this depart-

[1] *Transactions of the American Society of Mechanical Engineers*, vol. xxiv., pp. 1337 to 1456 (1903).

ment consist of one or more specially-trained men for each branch of the industry. For example, one for the mine and one for the mill, the balance of the department to consist of the superintendents of the respective departments and the manager of the company—the superintendents and the manager not to take the initiative, but to work with the men who have had special training in each line.

By making use of a card-system in which the workman fills up his own card daily, to be approved by his foreman, it is often possible for the planning-department to handle all its office-work without extra help. The offices of this department should be situated as close to the work as possible, in order to keep in constant touch with its progress.

In starting work, each workman passes through the planning-department and secures his daily card, on the back of which the planning-department has designated plainly and exactly what his day's work is to be, and the tools he will require.

It hardly seems necessary to explain the advantages of such a department. It makes it unnecessary for the workman to think out his task or stop to hunt up tools, supplies, etc., as he is thus definitely instructed daily what to do, the tools to use and how to use them, leaving nothing for him to think of but performing his task in the most advantageous way and receiving his high wages. The planning-department naturally develops a standard of efficiency, which if left to the foreman often suffers in all directions.

Mr. Taylor says in his paper that he has never been opposed by a strike, although he has been engaged for a greater part of his time, since 1883, in introducing this system of management in different parts of the country, and in a great variety of industries.

It certainly seems that a system which makes it possible for the workman to earn high wages should be advantageous in preventing labor-disturbances. This in itself would, in many cases, be a sufficient saving to the employer to warrant the adoption of the system, to say nothing of its possibilities in reducing operating-costs and increasing output.

This system is often regarded as applicable only to routine jobs; but it is so flexible that a daily task of several duties can be prescribed with decided success to one or more workmen,

Number of Hole.	Depth.	Location. *No. 3 Drift.* REMARKS.	Number of Hole.	Depth.	Location. *No. 3 Drift.* REMARKS.
1	*4′ 2″*				
2	*3′ 11″*				
5	*4′ 1″*				
6	*4′ 0″*				
3	*4′ 4″*				
4	*4′ 0″*				
7	*3′ 10″*	*Check.*			
8	*4′ 3″*				
8	*32′ 7″*	Total A. M.	*8*	*31′ 2″*	P. M.

Name of Driller, *John Peters.* Helper, *Jack James.* Date, *Nov. 4–11.*

Drill, *2 1-2″ Holman.* *Peters, 5.80.*

Total Footage in Shift, *63′ 9″*, *James, 4.80.*

Bonus 24′ @ 15c. $3.45.

Approved,

Shift, *7–4.* *Joe Hendy, Foreman.*

FIG. 1.—FORM OF TIME-CARD.

Time in *7 A. M.*

Time out *4 P. M.*

Use *Holman 2 1-2'' No. 42.*

3–4 piece sets steel.

Column rig, No. 42.

Load Holes with 4 sticks Giant, using 5' fuse each charge. Tamp holes.

DRILL HOLE DIAGRAM.

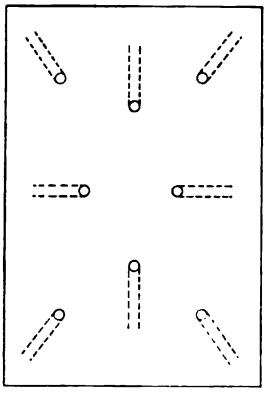

FIG. 2.—REVERSE SIDE OF TIME-CARD.

such as inspectors, janitors, millmen, yardmen, etc., whose work comprises many different functions.

Consider the usual work of a timekeeper, walking from place to place, and man to man, in all kinds of weather, spending only a few moments in each place, and trying to make an exact and detailed report on a separate card for each man. This could be done by each workman in a few minutes every day, thus giving the employer a most exact system of detailed work- and time-reports, while a timekeeper is only interested in the total time of work. A form of report-card is shown in Figs. 1 and 2.

It should also be the function of the planning-department to analyze mechanical and other operations. In a large concentrator-plant, one cannot but notice the network of belting. It is usually next to impossible to drive concentrators in any other way than by belting; yet how many mine-superintendents and managers realize that an analysis of the tension on belting shows a vast difference in power, efficiency, life and usefulness of the belt. It has been found by exhaustive tests that belts properly cared for by a trained workman will average twice the pulling-power, and require much less care, than when handled according to the usual practice in the concentrators I have observed.

I have been especially impressed with the fact that the majority of large working mines have, as a rule, extremely efficient and up-to-date hoisting-apparatus, embodying not only the best safety-appliances, but operating economically; and yet in most of these high-class installations, when run by steam, there is a great disregard for the steam-plant design and equipment. Considering that the cost of any kind of steam-fuel is usually higher in a metal-mining district than almost anywhere else, we may well be surprised that there is less regard paid to its value in such places than in coal- and oil-camps where fuel is very cheap.

There are few mines, using more than a 100-h-p. boiler-plant, where it would not be possible to evaporate from one-fourth to two-thirds more water per pound of fuel burned than is now evaporated, and do it with less labor-expense.

Assuming $4 per ton as the cost of a fair grade of bituminous coal delivered on the grates of a boiler-plant of 500 h-p.

rating, the saving of one-fourth of the fuel-expense means an annual saving of approximately $12,600. It is hardly possible to estimate the total value of such a saving (or the corresponding present waste) to the industry as a whole.

Space will not permit the consideration here of each phase of the mine- and mill-equipment; but what has been said of the possible value of improved apparatus and methods applies to nearly every branch of the industry and its relation to the results obtainable through a properly-organized efficiency-planning department.

In introducing this system, one cannot expect to attain the maximum output of the workmen at once; they gradually increase their working-capacity as they are taught efficient ways of handling themselves and their tools, and become accustomed to the new system.

In changing to this system I would advocate a start along the following lines:

1. The selection of a man of wide experience, not a miner, geologist, or mill-operator, but a man who has especial ability to observe what is actually going on, to take part in it, and to win the confidence and help of all the workmen.

2. The standardization of tools and methods throughout the mine, mill, and office.

3. The scientific study of time required to do unit-work in all lines.

4. The establishing of a card-system by which the workman can transmit the required information to the planning-department.

5. The making of a complete analysis of each piece of equipment used, and establishing a graphic-chart system of its operating-efficiency.

In making time-observations it is advisable to use a stop-watch, noting each observation and leaving nothing to memory. It is also necessary to record each detail of the operation, as a time-account of the total job is of little assistance to a first-rate planning-department.

In taking time-observations of machine-drill operation, I would advocate subdividing the time-observations into: (1) time required to place drill-column; (2) time to mount drill on column-arm; (3) time to make drill-connections ready for start-

ing; (4) time to crank machine forward; (5) time to put down first bit; (6) time to withdraw bit; (7) time to place new bit; (8) time to crank to drilling-position; and so on through the operations of a shift.

It may seem unnecessary to go into such detail; but if a hundred observations of this character were taken, it would enable the planning-department to fix the exact time for many other complete jobs of a different nature and position. The observation of each step is more simple than when a combination of several steps is taken together. A large part of the work comprises certain operations of standard duration, such as cranking a drill forward or back, or removing its steel. These standards, once determined, have great latitude of application.

In a great many mines, mucking is often the cause of serious delay to drillers and general progress. This can be obviated by having the trammers help clear the space required for the drill-column, and letting the drillers stand on the pile while putting in the upper holes, the muckers in the meantime working from the outside of the pile. By the time the upper holes are in, the drillers can work to advantage on the lower holes without any special support, while the muckers finish clearing the pile.

In timing the mucking-operation, observations should be taken about as follows: Loading shovel-pan and straightening up; conveying to dumping-position and dumping; returning to loading-position; load on shovel in pounds. If we let x equal time in minutes for loading shovel and straightening up; y, the time for conveying to dumping-position and dumping; and z, the percentage of time per shift spent in resting, then the time in minutes required to shovel 1 ton equals

$$\left[(x + y)\frac{\text{Pounds in ton}}{\text{weight of shovelful in pounds}}\right](1 + z).$$

Accurate analyses of the time-elements in this class of work make it possible to arrange economically the work in a mine so that mucking can keep pace with drilling- and other operations. Similar observations of time-elements will apply to a wide range of work usually classed under "miscellaneous jobs," in which no particular department-head has any special interest, and which, in general, is largely left to the handy man of the gang.

The principal objection which may be raised against the introduction of this system is the possibility of incurring the opposition of organized labor. But there are many considerations to the contrary. In the first place, we are to select first-class and intelligent workmen with whom to start the system, or, in other words, in introducing the system, one or two first-class men, carefully selected, are put on the new basis unknown to the other workmen. These men soon learn their earning-value, and have no desire to join a union of workmen below their grade.

To secure the greatest harmony between workmen and employer, the employer must be fair, fearless, and democratic. I do not believe that I have ever seen friction in any establishment in which the employer met his men half way and took the trouble to listen to their side of all matters. No manager ever lost any prestige through being well liked by his men.

By arranging a fixed plan through which each and every workman can in some degree express any private opinion he may have, labor-unions and strikes can be prevented. These suggestions and opinions are often of great value to the employer.

It is often possible for the management of larger concerns to organize a workman's club, which can meet periodically, and often with company officials, for written or oral discussions on subjects of interest to workmen. They usually appreciate lectures by trained men on improved methods of working, as well as an opportunity to read current literature or play clean games. While I do not believe in luxurious club-houses and conservatories, I do believe a well-organized workmen's club, provided with facilities for intellectual and bodily improvement, results in great contentment of workmen as a class. These, of course, should follow, and not precede, scientific management.

Such clubs replace entirely any desire on the part of a first-class workman to organize or join a labor-union. Let each employer organize in such a way as to make his men more contented, thereby, than with any outside inducements possible.

I realize that as long as an employer tries to drive his men, and at the same time pay them as poorly as possible, so long will there be demands for labor-unions or for other things of a rebellious nature.

The greatest success is to be obtained by leading workmen to better conditions; never by driving them. This does not mean that they are to have favors lavished upon them. It is only necessary that the employer have the whole interest of his workmen at heart—a spirit which is unconsciously radiated and felt by every one, and in exchange for which workmen will accomplish almost any task set before them.

It seems to me that the ultimate means of obtaining harmony of employer and employee is such a system of education as I have outlined, under which the workmen will expand their thoughts into actual thinking ability. While this would be a benefit to present workmen, I would go a step further and advocate the introduction into schools and colleges, and especially technical colleges, of a course of study dealing with advanced ideas on labor-problems and scientific management.

As I have already said, I believe the mining industry is greatly in need of more scientific organization and management; and I hope others will be led to investigate this subject more fully, with this end in view.

Mining-Methods at Nacozari, Sonora, Mexico.

BY D. C. LIVINGSTON, MOSCOW, IDAHO.

(Presented by invitation at a meeting of the Spokane Local Section of the Institute, February, 1912.)

THE Pilares de Nacozari mine is located in Sonora, 75 miles south of Douglas, Ariz. The town of Douglas is on the International Boundary and is the place at which the ores from the Bisbee mines are smelted. The Moctezuma Copper Co. owns the mines at Nacozari, and the copper-concentrates shipped to the Copper Queen smelter, at Douglas, make a good smelting-mixture with the Bisbee ores. The Moctezuma Copper Co. and the Copper Queen smelter at Douglas, as well as the railroad from Douglas to Nacozari, are owned by the Phelps, Dodge & Co.

The ore-deposit occurs in the form of a large ellipse with a major axis of approximately 2,000 ft. and a minor axis of about 600 ft., the major axis bearing about 9° W. of N. The

whole of this ellipse is more or less mineralized; the surface being principally an iron gossan with some occasional copper-stains. Below the oxidized zone, which is not more than 50 ft. deep, the minerals are pyrite and chalcopyrite, with occasional seams of chalcocite, the latter being rare. There appears to be no well-marked zone of secondary enrichment, the oxidized gossan over the greater part of the deposit changing suddenly to what is apparently the original unaltered ore. The copper-values are concentrated around the perimeter of the ellipse, and it is principally around this perimeter that the mining is done. The ore mined averages a little more than 3 per cent. of copper, with a small amount of silver, less than an ounce per ton. Some ore of considerably higher grade than this has been shipped, however, but the above average is of the mill-run. The ore is wider near the two ends of the ellipse than along the sides, and is mined in some cases to a width exceeding 200 ft. The country-rock for the first vertical 500 ft. is a volcanic acid breccia, probably rhyolitic; below this point it is more a basic breccia, probably andesitic. The contact is plainly marked on the surface by bluffs in one place, but it is almost impossible to trace it underground on account of the similar appearance of the two rocks when freshly broken. The change in formation has no apparent effect on the ore; if anything, it is somewhat richer but more bunchy in the lower formation, but work enough to verify this statement has not yet been done. The contact between the mineralized rock of the ellipse and the unmineralized country-rock is plainly marked by a more or less definite wall, the rock on both sides of this wall, however, being the same. The ore is extracted from this wall towards the center of the ellipse until it becomes too low-grade to mine, although the whole of the interior of the ellipse would run almost 1 per cent. of copper.

This property presents a very interesting study of the origin of ore-deposits and was being examined and mapped last summer for the Moctezuma Copper Co. by J. M. Boutwell, formerly of the U. S. Geological Survey. There is a similar deposit at Mount Morgan, Australia, which also is in the form of an ellipse, though less than half the size of the one at Nacozari. I believe it is worked chiefly for the gold-values, and the whole of the ellipse is more or less workable. At the

Braden copper-mine,[1] Chile, there is an ore-body of this same shape, but there the ore occurs along a contact of a volcanic neck which is intrusive in an older rock and different, while at Nacozari the country-rock and the interior of the ellipse are apparently the same.

The mine is opened by two 3-compartment shafts, situated one at each end of the ellipse, and an adit tunnel about a mile long which taps the deposit at a point about 600 ft. below the collar of the shafts. This adit is large enough to allow narrow-gauge, 25-ton railroad-cars to load from the mine-chutes, and it is chiefly due to this cheap handling of the ore that the enterprise has become so commercially successful. The railroad runs to the top of the bins above the concentrator, which is situated on the Nacozari railroad at the town of Nacozari, about 5 miles from the mine. From here the concentrates are shipped to the smelter at Douglas, Ariz. The daily output of ore is between 1,500 and 2,000 tons.

The levels in the mine are 100 ft. apart; and the general plan is to use every alternate level as a stoping-base, the intermediate levels being used for development, ventilation, and the distribution of waste to the stopes that are being filled. At the present time practically all the ore is being mined above the adit-level, which is known as the "700," and at the present rate of output there should be, above this level, on a rough estimation, an ore-supply for at least 10 or 12 years. The shaft at the north end has been sunk to the 1,000-ft. level, and some work has been done on this level, as well as on the 800- and 900-ft. levels.

Owing to the scarcity of timber in the neighborhood, and to the high price of imported timber, it has been necessary to adopt mining-methods which would eliminate the use of timber as much as possible. The nature of the rock is such that while it is not hard to drill or blast, it stands without timber in a most remarkable manner in most parts of the mine. This characteristic of the rock has also prevented the use of the caving- and top-slicing methods, which are being used so successfully at Cananea in some of the mines. Accordingly, some system of underhand stoping had to be devised, and the following two methods are used:

[1] *Trans.*, xl., 743 to 746 (1909).

(1.) The old method, Fig. 1, which is room-and-pillar with waste-filling as the work progresses.

(2.) The new method, Fig. 2, which is shrinkage with waste-filling after all of the ore has been extracted.

(1.) With the room-and-pillar method the mine-plan is laid off into pillar-lines at right angles to the strike of the ore and placed 50 ft. apart, dividing it into alternate stopes and pillars. A main working-drift is run approximately parallel to the strike, and cross-cuts are turned off every 100 ft. in the pillars. From these cross-cuts right-angle headings are turned off, and every 50 ft. run to the center of the stope and from these the sill-floor is cut out to the limits of the ore and to the pillar-lines, making the stope 50 ft. wide. The roof, where necessary, is supported by props or cribs, and a 6-ft. cut is then taken upward with stoping-drills, the blasted ore being drawn off immediately, and filling run in from the level above through winzes to within 4 or 5 ft. of the roof. The short drifts from the cross-cuts are timbered through the filling, and cribbed chutes and man-ways are carried up from the ends of these. In a stope 150 ft. long, there would be three of these chutes and man-ways.

There were no pillars being extracted when I was there, and it was probably the apparent difficulties of removing these pillars without losing a large amount of ore by mixing with waste, which led to the adoption of the new method.

(2.) With the shrinkage method, designed by C. A. Smith, the use of timber is almost entirely eliminated, as not even timbered chutes are used. Cross-cuts are turned off from the main haulage-drift every 50 ft. parallel to the pillar-lines, and extending to the limits of the ore.

The next step is driving what are called the "stopes," which are breasts 15 ft. wide and the height of an ordinary drift. These are driven parallel to the cross-cuts, and midway between them, and are carried to the limit of the ore. From the cross-cuts "shovel-ways" are turned off at right angles at intervals of from 20 to 25 ft., and are driven to connect with the stopes. Turn-sheets are placed at the mouths of the shovel-ways. The ground is now ready to commence underhand stoping. The stopes are carried up the same width (15 ft.) with stoping-drills to a height of 20 ft. above the floor, and are

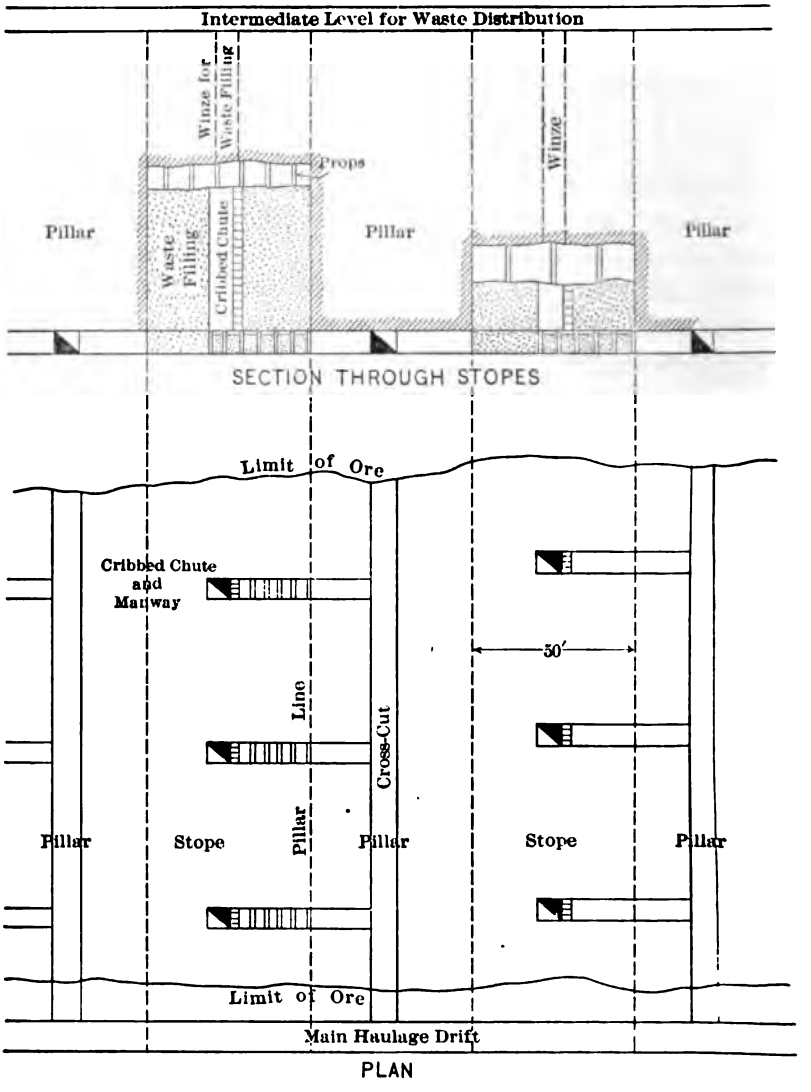

FIG. 1.—ROOM-AND-PILLAR SYSTEM OF MINING AS OPERATED AT THE PILARES DE NACOZARI MINE.

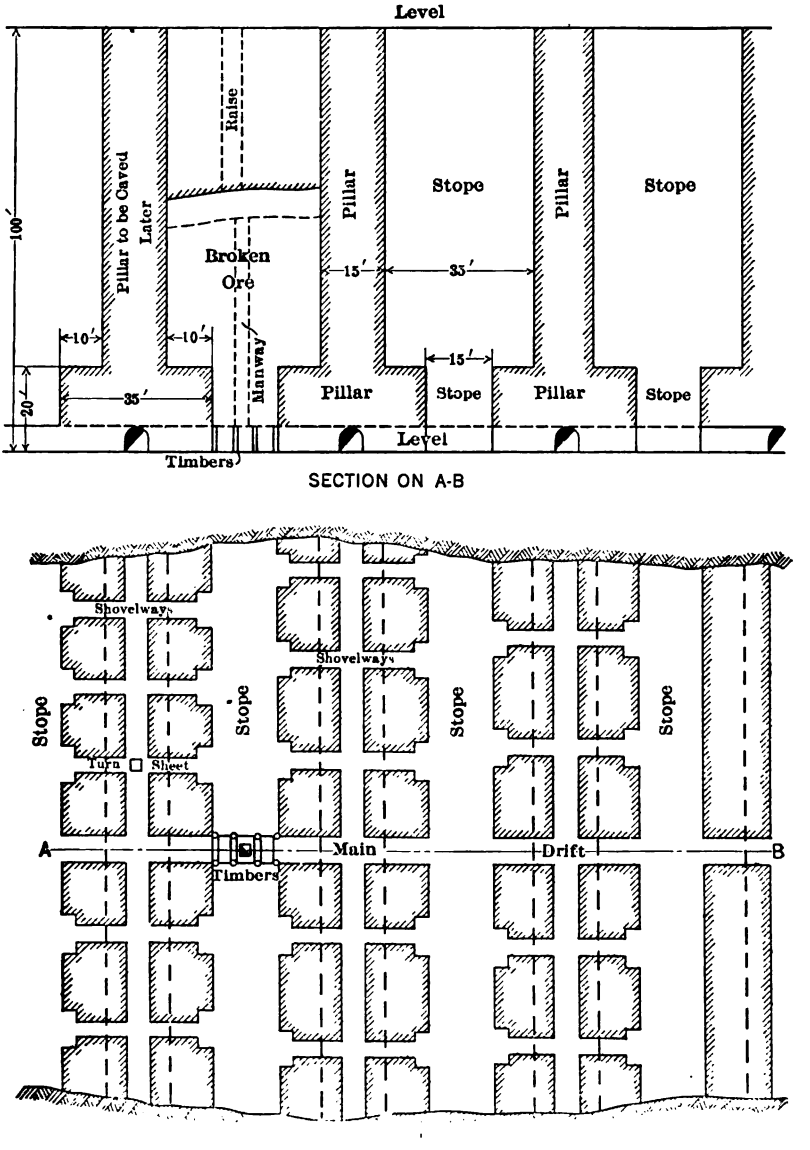

FIG. 2.—SHRINKAGE SYSTEM OF MINING AT THE PILARES DE NACOZARI MINE.

then widened 10 ft. on each side, making them 35 ft. in width, with 15-ft. pillars in between.

The machines are set up on the broken ore, enough being drawn out through the shovel-ways to keep a working-clearance. The stope is continued up in this manner to the next stoping-level. This work was started last summer, but it is evident that the main drifts will have to be timbered where the stopes cross them, in order to keep the broken ore off of the track, and cribbed man-ways built up through the broken ore at these points, but aside from this, with the exception of an occasional prop, no timber at all is used. There would be no absolute necessity for man-ways through the broken ore, as the stope can be entered by means of the winze from the level above.

The shovel-ways, which are merely rock-chutes, have been used in the mine for a number of years in the narrower portions of the vein instead of timber chutes. The reasons for their adoption are:

(1.) No timber required.

(2.) Ease and safety in blasting large boulders, which can be more safely and easily blasted in the shovel-ways than in the stopes, and without interfering with the work.

(3.) All work, except timbering, is done by contract with the natives, and it was found that unless all the rock was broken in the stopes sufficiently small to pass a chute readily (a difficult and expensive item in ground that breaks big where shrinkage is employed), it was just as cheap to have it shoveled into the car as to draw off from a chute that was continually hanging-up, and the expense of placing the timber was also eliminated.

In mining the pillars in the shrinkage system, I believe the plan is to put up raises in the pillars and from the raises to shatter the pillars with heavy shots, so that the pillars will cave when the ore is drawn out of the stope. If this method works out as expected at least 95 per cent. of the ore should be recovered.

As regards the comparative costs of the two systems: stoping in the wide stopes by the shrinkage system had not progressed sufficiently in the summer of 1911 to give a complete comparison, but what figures were available pointed to a considerable

saving in the shrinkage over the room-and-pillar system. This should be the case after the sill floor has been opened up, because, in the shrinkage system, there is no rock handled in the stopes until the ore is drawn out, whereas, in the room-and-pillar system both the ore and waste filling have to be handled in the stopes in wheelbarrows.

Replaceable Lips for Elevator-Buckets.

BY H. J. MAGUIRE, WALLACE, IDAHO.

(Presented by invitation at a meeting of the Spokane Local Section, February, 1912.)

THOSE familiar with mill-practice understand the work required of an average bucket-elevator, but I wish to call special attention to the wear on the buckets. I have been studying in what manner the life and service of the bucket might be improved, and I feel that an adjustable lip, attachable to each bucket without the aid of rivets, will be of material advantage.

For illustration, consider the feed-elevator of a typical mill in the Cœur d'Alène district. The ore is carried by water in a launder from the feed-rolls and dumped in the "boot" of the elevator, directly in front of the ascending bucket. Middlings from the return-rolls are similarly handled. Under this condition the bucket not only carries the ore, but also picks it up, and this latter requirement determines the life and service of the bucket. It might be said that it would be better practice to convey the feed to the elevator and dump it directly into the bucket, but at one mill at which this plan was tried the results were disappointing and the rubber belt was badly worn.

The wear of the bucket is confined to the lip and the corners which have to pick up the material. When the lip on a corner of a 7- by 12-in. malleable cast-iron bucket is worn down about 3 in., the bucket is of no further value as a carrier and has to be replaced. The worn bucket, containing about three-quarters by weight of the metal of the original bucket, has a scrap value of about 10 per cent. of that of a new bucket. To the cost of the bucket, however, should be added the time-

cost of replacing the old bucket by a new one. This time-cost is considerable when the loss per hour, to cease operating the mill, is about $200.

It is out of the question to put lips on a bucket unless the latter was originally fitted with this device. For if it is attempted to fit lips upon buckets not originally provided with them, it will be necessary to drill holes for rivets and then spend considerable time fitting the lips to the irregular contours of the worn buckets. The improvement suggested is an adjustable attachment which can easily be fastened to, or removed from, the base of a bucket without the use of rivets. This attachment, to be made of a high-grade steel drop-forging or casting, will combine toughness, hardness, and tensile strength. The base of the bucket may be a cheap casting,

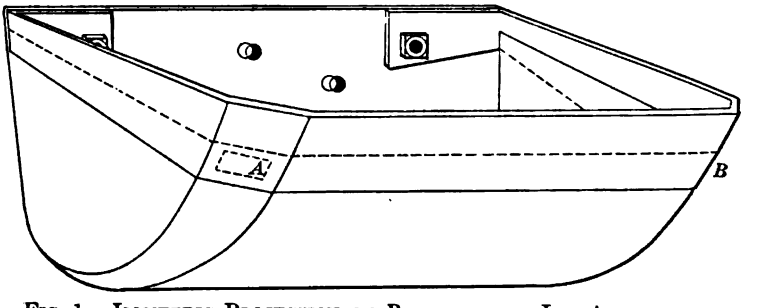

FIG. 1.—ISOMETRIC PROJECTION OF BUCKET WITH LIP ATTACHMENT, SHOWING THE LIP BOLTED TO BUCKET.

possibly ordinary cast-iron. This combination of lip and base will not increase the cost of the improved bucket much, if any, above that of the ordinary bucket now in use, but will increase the life of the bucket, doing the same amount of work, about 400 per cent., with a corresponding decrease in time lost in changing buckets.

This improved bucket is illustrated in Fig. 1, which shows a projection of the bucket with the attachment in place. Two bolts, one on each side of the elevator-belt, pass through the base and attachment and hold both fast to the belt. These end-bolts are 0.25 in. longer than the others. Fig. 2 shows the bucket with part of the lip removed, illustrating the manner in which it is fitted to the base and the opening at *A* which receives the lug *A*. There is another opening at *B*, which is not shown.

Probably a more desirable arrangement than this would be to place the attachment inside the base, in which case the outside of the lip and the inside of the base would be rabbeted.

I am indebted to Stanley A. Easton, General Manager of the Bunker Hill & Sullivan mine, for the following data, which bear out my statement, and show that others have been trying to improve the service of elevator-buckets. He says that they use there 7.75- by 17-in. malleable cast-iron buckets, which give approximately 80 days' continuous service on middlings-eleva-

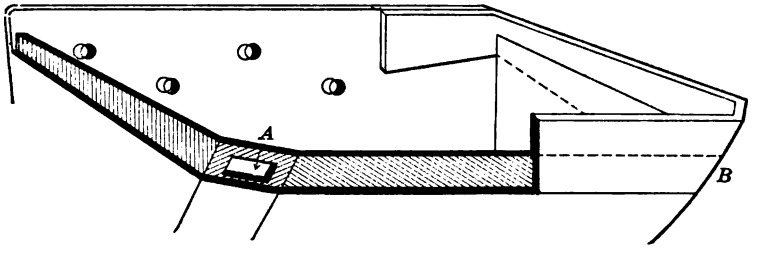

FIG. 2.—BUCKET WITH PART OF LIP REMOVED, SHOWING THE SHOULDER ON WHICH IT RESTS; ALSO THE OPENING AT *A* WHICH RECEIVES THE LUG *A*, FIG. 1. ANOTHER LUG AT *B*, FIGS. 1 AND 2, IS NOT SHOWN IN THE DRAWINGS.

tors, in the old mill, handling 350 tons per day. Placing a manganese-steel lip on one bucket in a set of 72, added 35 days' service to the 80 days for the whole set. This set then gave a total service of 115 days. The cost of buckets per year, for 1,300 lin. ft. of belt handling 2,750 tons per day, is $1,436.80.

On this basis, the total cost for elevator-buckets in the concentrators of the Cœur d'Alène district must be at least $10,000 per year. The saving by installing improved buckets can easily be estimated.

Ore-Treatment at Republic, Wash.

BY FRANCIS A. THOMSON,* PULLMAN, WASH.

(Presented at a Meeting of the Spokane Local Section, February, 1912.)

BUT little has been written concerning the camp of Republic. In 1900 Chatard and Whitehead[1] reported the results of some experiments with samples of ore from the Republic mine, and a few years later Fritz Cirkel[2] described the construction, operation, and results attained in the mill at the Mountain Lion mine. From time to time brief articles, mostly of a semi-technical nature, have appeared in the *Engineering and Mining Journal* and other mining periodicals. The general geology of the camp, especially in regard to its areal features, has recently been ably described by Joseph B. Umpleby.[3]

To bring to mind the general features of the district it may perhaps be well to review briefly some facts in regard to it, before discussing the treatment of the ore.

I. *History and Production.*

The town of Republic is situated in the NW. corner of Ferry county, about 25 miles south of the International Boundary, on the head-waters of the San Poil river, in what was formerly the "north-half" of the Colville Indian reservation.

On Washington's birthday, 1896, this portion of the reservation was thrown open to mineral location, and within a few weeks important discoveries had been made. The present Eureka mining-district was organized, the camp grew apace, and consignments of ore were shipped by wagon to Grand Forks, 30 miles away; or to Marcus, a somewhat greater distance.

It is reported that the first 1,500 tons shipped from the Republic mine netted the owners $160 per ton, after being hauled

* State College of Washington.

[1] *Trans.*, xxx., 419 (1900).

[2] *Journal of the Canadian Mining Institute*, vol. v., pp. 274 to 287 (1902).

[3] The Geology of the Republic District, *Bulletin No.* 1, *Washington Geological Survey* (Olympia, 1910).

60 miles by wagon and 300 miles by rail. Prior to the advent of the railroads into the camp, this mine paid in dividends practically $400,000.

An exact estimate of the production of the district is difficult to secure. Table I. has been compiled from all available data.

TABLE I.—*Production of Republic Camp.*

Classified Production from Jan. 1, 1896, to Dec. 31, 1905.

Property.	Disposal of Ore.	Tons.	Approximate Average Value.	Total Value.
Republic	Milled	24,000	$17.90	$430,000
	Shipped	13,000	80.00	1,040,000
Quilp	Shipped	24,428	10.76	262,800
Mountain Lion	Milled	12,000	10.83	130,000
	Shipped	8,700	7.00	60,900
Lone Pine	Shipped	8,000	14.00	112,000
Ben Hur, California, Morning Glory, Tom Thumb, Blacktail, El Caliph, San Poil, Insurgent (In order named.)	Shipped	7,000	21.00	147,000
Total for ten years, ending 1905				$2,182,000

Unclassified Annual Production.

Year.	Tons Ore.	Gold.	Silver.	Total.
1906	6,223	$69,957	$17,653	$87,610
1907	450	4,032	4,700[a]	8,732
1908	584	15,343	4,190	19,530
1909	11,299	209,114	27,817	236,931
1910	33,781	713,604	100,082	813,686
1911	24,500[a]	475,000[a]	75,000[a]	550,000[a]
Totals	76,857	$1,487,050	$229,442	$1,716,492

[a] Estimated.

Combining the total of classified production prior to 1905 with the sum of the annual production since that date, we get for the value of the total production to Jan. 1, 1912, $3,898,492, which I believe is very close to the truth.

An examination of Fig. 1, in which the gold-silver production of the district and of the whole State is plotted, shows very clearly the successive stages of the camp's prosperity, which

continued unabated until the middle of 1901, when the large mill of the Republic Power & Cyanide Co. closed down.

In 1903, with the arrival of the railroads, there was a sporadic activity, which died as soon as the smelting market had been glutted with the siliceous ore, in addition to which of course most of the ore available was of too low a grade to stand a combined freight- and treatment-charge of $10 per ton, which was only lessened in the case of large contracts. Since 1909 the camp has again become vitalized, stimulated largely by the activity of J. L. Harper and his associates. Beginning with the opening of a previously overlooked (probably intentionally) rich spur-vein in the old Republic mine, activities have spread to

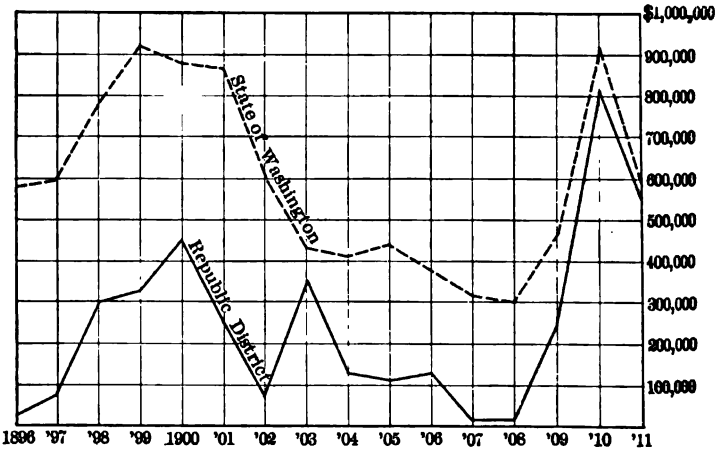

FIG. 1.—GOLD- AND SILVER-PRODUCTION OF THE REPUBLIC DISTRICT AND THE STATE OF WASHINGTON FROM 1896 TO 1911.

the Surprise, Knob Hill, Blacktail, Lone Pine, San Poil, Ben Hur, Quilp, and other properties.

Up to the present time, however, this activity has been pernicious from the point of view of good mining. Many of the properties have been operated as though by leasers, with the inevitable consequence that, to use the vernacular, "the eyes are being gouged out of the mines," only that ore being shipped which will show a profit after paying a combined freight- and treatment-cost of from $8 to $10 per ton. This means that everything below $12 per ton is left in the stope, on the dumps or is not broken at all. These unfortunate conditions will continue until successful local treatment is available.

Republic is unusually fortunate in transportation-facilities. Two lines, the Washington & Great Northern and the Kettle Valley & British Columbia railroads, enter the camp, and almost every property, no matter how insignificant, has its own spur from one or both of these roads.

II. *Geology.*

The geology of the camp, from a mining engineer's point of view, is extremely simple. The veins are fissure-fillings, with considerable wall-replacement at various points; all are inclosed in a series of tertiary eruptives and eruptive breccias, grading from andesite to rhyolite. According to Umpleby, the ore-bodies are genetically related to one of the latest of these flows. In general, the veins strike almost due N–S., varying a few degrees to the east and west; usually the dip is to the eastward at a high angle. In width they vary from 10 in. to 10 ft., usually from 2 to 5 ft., however, with an average of about 4 feet.

The prevailing ore of the camp is a fine-grained, close-textured, chalcedonic quartz, the values consisting entirely of gold and silver. Free gold is frequently visible in the high-grade specimens; free silver occurs, probably as an oxidation-product, in the upper portions of many of the veins. In the average grade ore, however, neither of these metals is visible.

The most remarkable and characteristic feature of the ore is the unique banding, due to crustification. Indeed, the appearance of the ore is so characteristic that any one at all familiar with the district can identify a piece of Republic ore at a glance. This crustification, or banding, is made evident by alternations of fine- and coarse-grained quartz, together with psuedo-bands of metallic mineral. These "metallic bands" are usually so indistinct that they have the appearance of pencil-streaks on the under side of a ground-glass plate. Further examination heightens this resemblance, for under a powerful lens the streaks do not appear continuous, but are seen to be made up of numerous little points arranged along a continuous line. The determination of the exact character of these metallic particles is a problem of some importance in connection with the treatment of the ore. Various attempts have been reported in this direction, none of them with any positive results.

About four years ago I became interested in the problem of treating this ore, and at various times I have worked upon it, both personally and with the aid of my students and assistants, and while I am not yet ready to pass positively on the entire subject, the results obtained are interesting.

In the first place, it became evident early in the investigation that the only hope of finding the exact character of the metallic minerals lay in the examination of the high-grade specimens. This, of course, is open to the objection that the minerals in the high-grade ore may be different from those in the low-grade. From examination of a large number of samples from various parts of the camp, however, we have come to the conclusion that the difference is one of degree, rather than of kind, the minerals in the high-grade ore being identical with those in the low-grade, but present in greater quantity.

One of the first samples examined consisted of a piece of high-grade from the spur-vein in the Republic mine. This specimen, examined under the microscope by reflected light, showed particles of gold, in part apparently crystalline, and a few small cubical crystals of pyrite, with marked striations, scattered throughout a more abundant steel-gray mineral of metallic luster, devoid of crystalline form. A few crystals resembling fluorite were noted. (Recently specimens have been secured in which fluorite is abundant.)

An analysis of this high-grade specimen gave: SiO_2, 75.7; Al_2O_3, 2.8; Fe, 1.0; Sb, 6.6; Cu, 9.1; S, 3.2; Au, 0.3; Ag, 1.1; total, 99.8 per cent.

If we discard the non-essentials, silica, alumina, and gold, and ignore the extremely small amount of pyrite visible, we get, by recalculating the remaining elements, the figures shown in the first column of Table II. The data in the last two columns of this table afford a comparison between this mineral and the tetrahedrites listed by Dana.[4] The sulphur is 4.2 per cent. below the recorded minimum, the antimony 1.2 per cent. above the recorded maximum; but in a mineral showing such wide variations in composition, this does not seem an insuperable difficulty, and, pending correction by the mineralogists, I shall call it "tetrahedrite."

[4] *A System of Mineralogy*, 6th ed., p. 139 (1892).

TABLE II.—*Comparison of Republic Mineral and Tetrahedrite.*

Element.	Republic Sample.	Tetrahedrite Analyses (Dana).	
		Maximum.	Minimum.
	Per Cent.	Per Cent.	Per Cent.
S	15.4	29.60	19.66
Sb	31.4	30.18	9.06
Cu	43.3	44.08	10.8
Fe	4.8	13.08	
Ag	5.2	31.29	

This specimen was sufficiently rich to sweat little bubbles of gold when roasted. It was handed to me as a sample of gold telluride (sylvanite?), its behavior on roasting being cited as evidence of its telluride character. As an actual matter of fact, it contained no tellurium at all. Selenium, if present, escaped detection.

From time to time other high-grade specimens have been examined with substantially the same results, except that chalcopyrite has been added to the list of possible minerals. Recently we received some extremely rich samples from the 100-ft. level of the Knob Hill mine. These samples, in which free gold is present in great abundance, show a comparative absence of the gray copper-mineral. Chalcopyrite, however, is quite abundant, as is also cerargyrite. In places minute sparkling black crystals of pyramidal habit are abundant, which can only be detected under the microscope and are with difficulty made available for testing. The work of investigating these crystals has been referred to Roswell E. Sampson,[5] Assistant Professor of Metallurgy, State College of Washington, who reports the extremely rare metal selenium as being abundant; silver also is present in considerable amount; antimony is suspected. (Provisionally, it might be suggested that the mineral is perhaps a selen-antimonide of silver; the antimonial analogue of the hypothetical mineral, rittengerite, of Dana.)

It is suggested by Mr. Umpleby, in his report, that probably a large part of the gold occurs as a gold selenide. Such a compound has not been reported from anywhere else, and the prominence of free gold in all of the high-grade specimens, even from the levels well below the zone of oxidation, would be against the supposition.

[5] Deceased Dec. 12, 1912.

Tellurium, in our investigation, has not been found.

The occurrence of selenium is unique and interesting; only in two other places, so far as I am aware, has it been found on this continent—at Goldfield and at Tonopah. It need, however, not cause any uneasiness to the cyanider, since selenium is extremely soluble even in the most dilute cyanide solutions, and it will therefore be unlikely to interfere with the solubility of the gold even if combined with it, which I seriously doubt.

III. Ore-Treatment.

From time to time in the history of the camp of Republic attempts were made to treat the ore on the ground, and the general impression appears to be abroad that all of the plants were unsuccessful. Such, however, is not the case.

In point of historical order, the plants may be presented as follows:

1. *The Plant of the Republic Gold-Mining & Milling Co.*—This plant used the Pelatan-Clerici process, following wet-crushing in ball-mills, and was reasonably successful in spite of the now universally recognized defects of the process. For some time, with its 50 per cent. recovery on high-grade ore, it was looked upon as quite an achievement. It treated, in all probability, 2,000 or 3,000 tons of ore. It is interesting to note that D. W. Rathfon, of Republic, has recently completed the re-treatment of the entire tailings-pile, from which a good profit was realized.

2. *The Barrel-Chlorination Plant of the Republic Reduction Co.*—This plant, from an inspection of its remains, appears to have operated scarcely at all.

3. *The Mountain Lion Mill.*—This amalgamation- and cyanide-plant began operation Mar. 15, 1900, and ran continuously till Nov. 1 of the same year, according to the flow-sheet shown in Fig. 2. It treated 12,000 tons of ore of an average value of $10.83 per ton,[6] and made a total extraction of 55 per cent.; 41 per cent. of this recovery being made by amalgamation and the balance by the cyanide treatment. The milling-cost per ton during this period is given as $3.73, which is quite a reasonable figure for so small a plant, remote from transportation. The mill is said to have cost about $60,000 and was

[6] Fritz Cirkel, *loc. cit.*

thoroughly well constructed from a mechanical standpoint. When we keep in mind, however, the extremely fine-grained character of the ore, it is not at all strange that cyanidation did not yield satisfactory results when applied to a Huntington mill product. An examination of the tailings from this plant shows that a large part of the loss was in contained particles—gold surrounded by a chalcedonic matrix.

(1) Shaft-Bins.
(2) Grizzly.
(3) 9- by 15-in. Blake Crusher.
(4) Gravity Tramway.
(5) Mill-Bins.
(6) Four 5-Stamp Batteries, 1,200 lb., 7-in. drop, crushing through 30-mesh.
(7) Copper Plates.
(8) Four 5-ft. Huntington Mills (80- or 100-mesh).
(9) Bucket-Elevator.
(10) Cone-Bottom Settler (12 by 12 ft.)
Pulp. Return Water, to (6).
(11) Conical Agitator (10 by 11.5 ft.). 8 hr. with 0.4 per cent. KCN ; 4 hr. with 0.2 per cent. KCN.
(12) Six Percolating-Tanks (Decanting and Filtering with vacuum). Tanks 24 by 4.5 ft.
Solution. Tailings.
(13) Zinc-Boxes.
Precipitate. Barren Solution.

FIG. 2.—FLOW-SHEET OF THE MOUNTAIN LION MILL.

In 1903 attempts were made to adapt the Hendryx electro-cyanide process to the needs of the plant, but presumably without satisfactory results. Some experiments in vanner-concentration were conducted at a later date with, I am told, a fair measure of success.

4. *The Plant of the Republic Power & Cyanide Co.*—This plant, locally known as the "Old Republic Mill," was erected on the lines of the plant of the Mercur Consolidated Co., by D. C. Jackling, now of the Utah Copper Co., to whom I am indebted for many of the details here presented in regard to its success.

The mill began operation in October, 1900, and was closed down in July, 1901, for various reasons, chief of which being the inadequate ore-supply, due to failure to keep the development-work ahead of the demands of the mill. Fig. 3 shows the scheme of treatment employed and it is scarcely necessary to enter into any further description.

(1) Crusher-Bins.
(2) No. 5 D Gates Gyratory.
(3) Trommel, 0.75 in.
Undersize. Oversize.
(4) 15- by 36-in. Rolls. (5) Gates Gyratory "H," to (3).
(6) Sample- and Storage-Bins.
(7) Two 5- by 36-ft. Driers.
(8) Two 15- by 36-in. Rolls.
(9) Two 15- by 26-in. Rolls.
(10) 40-Mesh Shaking-Screens (90 per cent. through 60-mesh, 55 per cent. through 200-mesh).
Undersize. Oversize, to six 30-in. Griffin Mills, to (10).
(11) Four Jackling Roasting-Furnaces. Hearth, 12 by 100 ft. (10 to 12 cords wood each per day).
(12) 12 Steel Leaching-Tanks, 22 ft. sq. by 6.5 ft. deep.
Treatment, 4 to 6 days with 0.5 per cent. KCN;
2 to 3 days with 0.3 per cent. KCN.
Solution. Tailings.
(13) Two "Gold" Tanks, 10 by 14 ft.
(14) Two Zinc-Dust Precipitation-Tanks, 8 by 12 ft.
Precipitate. Solution, to Sump.
Four 24- by 36-in. Johnson Presses.
Precipitate, to Bullion-Room. Solution, to Sump.

FIG. 3.—FLOW-SHEET OF THE "OLD REPUBLIC" MILL.

During nine months of operation this plant treated 21,240 tons of ore, averaging gold, 0.77 oz., and silver, 2.3 oz. per ton. The recoveries (by bullion), as shown in Mr. Jackling's report issued Aug. 1, 1901, were gold, 91.3; silver, 15.5; total, 85.05 per cent., the average cost during the last six months of operation being $4.65 per ton, exclusive of general expense, amortization, interest, etc. A steady, uninterrupted supply of ore sufficient to enable the mill to run at full capacity would have reduced this cost materially.

Of these four plants, the Mountain Lion mill is the only one in existence to-day.

The railroads in 1903, as has been stated, did not bring the expected revival, for reasons already discussed. Republic then lapsed into "innocuous desuetude," the mines were nearly all abandoned, and not a few of the more important ones, including the Republic itself, were sold for delinquent taxes. In the meantime, cyanidation had been making rapid strides, the tube-mill and the vacuum-filter had demonstrated their practicability and success in cyanide-work, and we began to wonder if Republic ores would not prove amenable to the new order of things.

(1) No. 5 Gates Gyratory.
(2) 50 Stamps, 1,500 lb., 6-in. drop, 110 per min., 4-mesh Screen.
Crushing in KCN Solution.
(3) Chilean Mills, 20- to 30-mesh.
(4) Classifiers.
Sand. Slimes.
(5) Tube-Mills (70 per cent. through 200-mesh).
Sand. Slime.
(6) Muntz Metal Amalgamating-Plates.
(7) Classifiers.
Slimes. Sands, to (5).
(8) Pulp-Thickeners.
Underflow. Overflow, to (11).
(9) Pachuca Agitators.
(10) Merrill or Oliver Filters.
Solution. Tailings.
(11) Clarifier.
Precipitate. Barren Solution.
(12) Zinc-Dust Precipitation-Plant.

FIG. 4.—FLOW-SHEET OF THE PLANT DESIGNED IN 1909.

In 1909 I undertook the investigation of the possible cyanide-treatment of Republic ores by standard methods of fine-grinding and filtration, with the results shown in Table III. At the same time I advocated a thorough sampling of the entire camp, with a view to the erection—should the tonnage and values justify it—of a plant to be operated in connection with one of the larger properties, as well as to provide custom-milling facilities for the

entire camp. It was expected that this plant would follow the general lines laid down in Fig. 4, subject, of course, to some preliminary work to be undertaken by altering one of the existing plants into a testing-mill. Owing to reasons extraneous to this discussion, the entire project was dropped; but the correctness of the mill-scheme has been amply demonstrated by the success of the little mill erected by D. W. Rathfon to treat the tailings from the old Pelatan-Clerici process. Mr. Rathfon's plant consists of a 5- by 22-ft. tube-mill, short Pachuca tanks, Oliver filter, and zinc-shaving precipitation-boxes, and has given better than 90 per cent. extraction, by bullion, on the tailings mentioned.

TABLE III.—*Treatment of Typical Samples of Republic Ores.*

Agitation with 0.25 Per Cent. KCN Solution.

Agitation.	Gold-Extraction.	Silver-Extraction.
	On 100-Mesh Material.	
Hours.	Per Cent.	Per Cent.
8	76 to 81	75
16	79 to 85	76 to 79
	On 150-Mesh Material.	
8	78 to 88	80 to 82
16	80 to 88	82 to 84
	On 200-Mesh Material.	
8	89 to 94	82 to 86
16	95 to 99	86 to 94

Amalgamation of 200-Mesh Material.

Gold-extraction, 31 to 37 per cent.	Silver-extraction, 8 to 10 per cent.

The plant of the North Washington Power & Reduction Co., on which work was begun early in 1911, is just commencing operation. It is following standard lines and should, under proper conditions, yield satisfactory results.

By the courtesy of H. W. Newton, metallurgist, under whose superintendence the construction of the plant has been carried on, I present, in Fig. 5, a flow-sheet of this mill.

(1) Main Storage-Bin.
(2) Shaking Grizzly.
Oversize. Undersize, to (4).
(3) 10- by 20-in. Blake Crusher.
(4) Shaking Grizzly.
Oversize. Undersize, to (6).
(5) 10- by 12-in. Dodge Crusher.
(To be replaced by 16- by 42-in. Rolls.)
(6) 4- by 6-ft. Trommel.
Oversize. Undersize, to (8).
(7) 14- by 30-in. Traylor Rolls.
(8) 16-in. Belt-Conveyor.
(9) Two Vezin Samplers (in series).
Discard. Sample to Assay Office.
(10) Mill-bin.
(11) 6-ft. Trent Chilean Mill Crushing in Cyanide Solution.
(12) Akins Classifier.
Sands, to (13). Slimes, to (15).
(13) 5- by 22-ft. Tube-Mill, El Oro Lining, Scoop-Feed, 28 rev. per min.
(14) Cone-Classifier.
Slimes.
(15) Four Trent Agitators, 33 by 20 ft.
Pulp. Overflow from No. 1, to (17) or (19).
(16) Box Classifiers.
Overflow. Spigot, to (18).
(17) Trent Disk Thickener.
Pulp. Solution, to (19).
(18) Oliver Filter, 11.5 by 16 ft.
Tailings. Solution.
(19) Clarifier.
(20) Trent Zinc-Dust Precipitation-Plant.
(21) Gould Triplex Pump.
(22) Perrin Press.
Precipitate. Barren Solution.

FIG. 5.—FLOW-SHEET OF THE PLANT OF THE NORTH WASHINGTON POWER & REDUCTION CO.

The San Poil Consolidated Co. also has recently undertaken the erection of a treatment-plant along the lines shown in Fig. 6.

In neither of these plants has any provision been made for amalgamation or concentration; and while I believe that satisfactory extraction may be made in this way, it seems, from our experiments, that a sufficient recovery is possible by amalgamation to make this simple addition to the mill-scheme fully worth while. The omission of concentration is, I think, correct; although I understand that the contemplated additions to the Rathfon mill, which are designed to fit it for the treatment of ore, as soon as the supply of tailings is exhausted, include an equipment of belt-vanners.

(1) Ore-Bin.
(2) 18-in. Belt-Conveyor.
(3) Williams "Hammer Trommel" Mill.[7]
(4) Elevator.
(5) Storage-Bin.
(6) 14- by 24-in. A–C Rolls "C."
(7) 5- by 22-ft. Gates Tube-Mill, Trunnion Feed, Iron Lining, Crushing in Cyanide Solution.
(8) Sand-Wheel.
(9) Duplex Dorr Classifier.
Slimes. Sands, to (7).
(10) 10- by 25-ft. Dorr Thickener.
Pulp. Overflow, to (12).
(11) Nine "Grass-Valley" Air-Agitators (12 ft. in diameter by 18 ft.) in series arranged for decantation.
Pulp. Solution, to (14).
(12) Cone-Bottom Agitator used as Wash-Tank.
Slimes. Overflow, to (14). Solution.
(13) 16-ft. Oliver Filter.
Solution. Tailings.
(14) Clarifiers.
(15) Zinc-Boxes.

FIG. 6.—FLOW-SHEET OF THE SAN POIL CONSOLIDATED MILL.

[7] (A squirrel-cage disintegrator, 3 ft. in diameter by 6 ft. long, making 100 rev. per min., in the same direction as a central shaft making 600 rev. per min., upon which are 32 16-lb. hammers slung from the shaft by chains. The space between the bars is $\frac{3}{16}$ in., and the mill is intended to reduce mine-run ore to this size.)

Republic is the best gold-camp in the State of Washington to-day, and there is no question that the ore can be satisfactorily treated on the ground for $2 or less per ton if cheap electric power is available. Conservatively, then, we may say that all ore more than $5 per ton in value can be handled at a profit. How much of this $5 ore is available is, of course, the question. Personally, I believe the amount is large, possibly running up into millions of tons; but this estimate is based upon general impression gained from numerous visits to the camp, supplemented by knowledge gained in sampling two or three of the larger properties, and not through sampling and measurement of the ore exposed in the different workings of all the various mines, without which, of course, no estimate of any value can be made.

The No. 2 Unit of the Mill of the Bunker Hill & Sullivan Mining & Concentrating Co.

BY R. S. HANDY,* KELLOGG, IDAHO.

(Presented by invitation at a meeting of the Spokane Local Section of the Institute, Wallace, Idaho, May 12, 1912.)

THE No. 2 unit of the West Mill of the Bunker Hill & Sullivan Mining & Concentrating Co., which went into commission Apr. 17, 1912, is structurally a twin of the No. 1 unit, which was started Nov. 9, 1909.

Up to the point where the ore enters the concentrating-department, it follows precisely the process described by G. Caetani in his article entitled, Milling of Lead-Silver Ores,[1] under the heading, Preparation of the Mill-Ore.

The process from this point forward, in the No. 2 section, is outlined on the accompanying flow-sheet. In describing it I shall attempt to analyze the products of each separate process. The constancy of the tonnage and the uniformity of the quality of the feed render this practicable.

No. 1.—550 tons of feed, all having passed 30 mm., is washed into two 15-mm. trommels, 4 ft. in diameter, 60 in. long, 17 rev. per min., 1 in. slope, ⅜-in. gauge steel.

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* Mill Superintendent Bunker Hill & Sullivan Mining & Concentrating Co.

[1] *Mining Magazine*, vol. ii., No. 5, p. 361 (May, 1910).

No. 1–A.—The oversize, 260 tons, is fed to two 4-compartment Harz jigs, 150 rev. per min., 1.5-in. stroke, 5-mm. sieves.

No. 1–A–1.—The products of these jigs are 25 tons of second-class concentrates from the first cup, @ 53 per cent. Pb; 80 tons of middlings @ 9.80 per cent. Pb; and 155 tons of tailings @ 0.40 per cent. Pb. The concentrates go to shipping-bins, the tailings to the dump; and

No. 1–A–2.—The middlings (including the hutch-product) are de-watered in a 3-mm. trommel, 24 in. in diameter, 30 in. long, 10 rev. per min., feeding (sample No. 1)

No. 2.—A set of 14- by 36-in. Chalmers & Williams standard rolls, spring tension, 75 rev. per min., set to crush to 0.5 in. in diameter. The hutch-products, having passed 5 mm. and including particles of rich ore, are not intended to be crushed, but are passed through the rolls for convenience in de-watering and conveying to the elevator. The discharge of the rolls, 110 tons (sample No. 2), is spouted at 45° angle to

No. 3.—A bucket-elevator, equipped with 16-in., 8-ply rubber belt, 117 ft. long; 90 malleable-iron buckets, 7.75 by 15 in.; speed, 494 ft. per min.; slope, 0.75 in. per ft.; top pulley 42 in. in diameter, bottom pulley 36 in. in diameter, gear-driven. This elevator discharges 355 tons to

No. 4.—2 trommels, 4 ft. in diameter, 17 rev. per min., 10 ft. long, consisting of 90 in. of 3-mm. 14-gauge screen and 30 in. of 7-mm. $\frac{3}{16}$-in. gauge screen.

No. 4–A.—The oversize, 200 tons, is fed to two 4-compartment Harz jigs, 175 rev. per min., 1-in. stroke, 3-mm. steel screens.

No. 4–A–1.—The products of these jigs are 2.3 tons of second-class concentrates @ 40 per cent. Pb; 160 tons of middlings @ 5.60 per cent. Pb; and 35 tons of tailings @ 2.30 per cent. Pb. Concentrates to bins, tailings to dump; and

No. 4–A–2.—The middlings (exclusive of the hutch-product, which goes to No. 7 rolls) are de-watered in a 2-mm. trommel, similar to the one already described, feeding (sample No. 3)

No. 5.—A set of C. & W. standard rolls, 75 rev. per min., set to crush through 7 mm. The discharge of these rolls, 160 tons (sample No. 4), is spouted at 45° to elevator, No. 3.

No. 1–B.—The undersize of 15-mm. trommels, No. 1, goes to

No. 6.—2 trommels, 4 ft. in diameter, 17 rev. per min., 10 ft. long, consisting of 90 in. of 3-mm. and 30 in. of 7-mm. screen.

No. 6–A.—The oversize, 90 tons, goes to two 4-compartment Harz jigs, 175 rev. per min., 1-in. stroke, 3-mm. steel screens.

No. 6–A–1.—The products of these jigs are 10.7 tons of second-class concentrates @ 53 per cent. Pb; 30 tons of middlings @ 5.3 per cent. Pb; and 50 tons of tailings @ 1 per cent. Pb. Concentrates to bins, tailings to dump, and the middlings (exclusive of the hutch-products, which go to No. 7 rolls) join No. 1–A–2 and go to No. 2 rolls.

No. 6–B.—The undersize of the 7-mm. screen on trommels No. 3, 50 tons, goes to

No. 6–B–1.—A 4-compartment Harz jig, 210 rev. per min., 0.75-in. stroke, 6-mesh brass wire screen.

No. 6–B–2.—The products of this jig are 2 tons of first-class concentrates @ 70 per cent. Pb; 1.5 tons of second-class concentrates @ 45 per cent. Pb; 20 tons of middlings @ 6.80 per cent. Pb; and 30 tons of tailings @ 0.90 per cent. Pb. Concentrates to bins, tailings to dump; and

No. 6–B–3.—The middlings (including the hutch-products of No. 4–A–2 and No. 6–A–1) are de-watered in a 2-mm. trommel feeding (sample No. 5)

No. 7.—A set of C. & W. standard rolls, 75 rev. per min., set to grind 80 per cent. through 3 mm. The discharge of these rolls, 85 tons (sample No. 6), is spouted with a grade of 3 in. per ft. to elevator, No. 3.

No. 6–C.—The undersize of the 3-mm. screens on trommel, No. 3, 100 tons, goes to

No. 6–C–1.—A de-watering box, 8 in. wide and 6 ft. long, with one pocket 4 in. deep and one pocket 12 in. deep, the slime overflow going to Esperanza classifier, No. 11. The pockets discharge through spigots to

No. 8.—Two Bunker Hill screens, 20 rev. per min., fitted with No. 136 ton-cap steel screen, 0.028-in. opening. The undersize (50 tons) goes to Esperanza classifier (No. 11).

No. 8–A.—The oversize, 50 tons, goes to two 4-compartment Harz jigs, 250 rev. per min., 0.25-in. stroke, 8-mesh brass wire screens.

No. 8–A–1.—The products of these jigs are 4 tons of first-class concentrates @ 79 per cent. Pb; 1 ton of second-class concentrates @ 60 per cent. Pb; 14 tons of middlings @ at 5.50 per cent. Pb; and 30 tons of tailings @ 1.70 per cent. Pb. Concentrates to bins, tailings to dump; and

No. 8–A–2.—The middlings are de-watered in a "V" settling-box and drawn off through a 1.25-in. spigot to (sample No. 7)

No. 9.—A 6-ft. C. & W. automatic adjusting Chilean mill, 25 rev. per min., fitted with 16-mesh needle-slot steel screens, discharging (sample No. 8) 64 tons to

No. 10.—A bucket-elevator with 16-in. 8-ply rubber belt 83.5 ft. long, 63 Salem pressed-steel buckets, No. 10 gauge, drilled with four $\frac{5}{16}$-in. holes in the bottom. Speed of elevator, 494 ft. per min.; slope, 1 in. per ft. Pulleys similar to those already described, belt driven. This elevator discharges into

No. 11.—An Esperanza classifier, 4 ft. wide, 20 ft. long, slope 4 in. per ft., with 42 red fir drags, steel armored; speed of belt, 17 ft. per minute.

No. 11–A.—Sand discharge, 147 tons, goes to

No. 12.—Two Bunker Hill screens, 20 rev. per min., fitted with No. 136 ton-cap steel screen.

No. 12–A.—Oversize returns to Chilean mill, No. 9.

No. 12–B.—The undersize, 137 tons, goes to

No. 13 —Two wooden Calumet hydraulic classifiers, with four spigots, discharging as follows:

Spigot No. 1.	Spigot No. 2.	Spigot No. 3.	Spigot No. 4.
60 tons to 4 Card tables, sample No. 9.	25 tons to 3 Card tables, sample No. 10.	16 tons to 4 Card tables, sample No. 11.	14 tons to 4 Card tables, sample No. 12.

No. 14–A.—7 Card tables (spigots Nos. 1 and 2), called "coarse" tables, make 4 tons of first-class concentrates @ 78 per cent. Pb; 5 tons of second-class concentrates @ 50 per cent. Pb, to bins; and 43 tons of tailings to tail-race.

No. 14–B.—9 Card tables (spigots Nos. 3 and 4), called "fine" tables, make 1 ton of first-class concentrates @ 78 per cent. Pb; 1 ton of second-class concentrates @ 60 per cent. Pb; and 17 tons of tailings @ 1.30 per cent. Pb. to tail-race. 8 tons of middlings from these tables @ 3.40 per cent. Pb go to

No. 14–B–1.—A Card table making 0.1 ton of second-class concentrates @ 38 per cent. Pb; and 8 tons of tailings @ 2.90 per cent. Pb to tail-race.

No. 14–A.—7 Card tables make 30 tons of middlings @ 4.12 per cent. Pb, which go to

No. 15.—A 2-in. Traylor centrifugal pump, 1,100 rev. per min., pumping to a "V" settling-tank, discharging through a 0.75-in. spigot to (sample No. 13)

No. 16.—A 6-ft. Hardinge mill, 22-in. straight cylinder, 32 rev. per min., silex lining, 3,000 lb. domestic pebbles, discharging 30 tons (sample No. 14) to No. 10.

Again taking up the jigging-system, on the middlings side:

No. 4-A.—The undersize, 80 tons, of the 7-mm. screen on trommels, No. 4, goes to

No. 4-A-1.—A 4-compartment Harz jig, 210 rev. per min., 0.75-in. stroke, 6-mesh brass wire screen.

No. 4-A-2.—The products of this jig are 1.2 tons of second-class concentrates @ 45 per cent. Pb; 65 tons of middlings @ 4.7 per cent. Pb; and 14 tons of tailings @ 2.40 per cent. Pb. Concentrates to bins, tailings to dump; and

No. 4-A-3.—The middlings join No. 6-B-3 to No. 7 rolls.

No. 4-B.—The undersize, 135 tons, of the 3-mm. screens on trommels, No. 4, goes to

No. 4-B-1.—A de-watering box same as No. 6-C-1, feeding

No. 4-B-2.—Two Bunker Hill screens, No. 136 ton-cap screen. The undersize, 56 tons, goes to the Esperanza classifier, No. 11.

No. 4-B-3.—The oversize, 80 tons, goes to

No. 4-B-4.—Two 4-compartment Harz jigs, 250 rev. per min., 0.25-in. stroke, 8-mesh brass screens.

No. 4-B-5.—The products of these jigs are 2.4 tons of first-class concentrates @ 75 per cent. Pb; 0.9 ton of second-class concentrates @ 60 per cent. Pb; 50 tons of middlings @ 4.3 per cent. Pb; and 27 tons of tailings @ 1.90 per cent. Pb. Concentrates to bins, tailings to dump, and middlings through de-watering box to Chilean mill, No. 9.

No. 11-B.—The slime discharge of the Esperanza classifier, No. 11, 68 tons (sample No. 15), joins the overflow of the Calumet classifiers, No. 13, 22 tons (sample No. 16), in the distributing-boxes of

No. 16.—Two "W" shaped thickening-tanks of the Old Dominion type, as described by Caetani, 30 ft. long, 9 ft. wide at the surface of the water, with ten 1.5-in. goosenecks discharging through spigots on each side of each tank.

No. 16-A.—Four spigots on each section (16 in all), with $\frac{3}{8}$-in. round opening and 36-in. head, discharge through collecting-launders and a 4-in. pipe, with 0.25 in. minimum grade, to 3 distributing-barrels feeding 40 tons at $\frac{1}{10}$ dilution to

No. 16-A-1.—Fourteen 6-ft. C. & W. frue vanners, 200 rev. per min., $\frac{5}{16}$ in. slope per ft., 4 ft. per min. belt-travel, making 4 tons of concentrates @ 50 per cent. Pb to bins, and 36 tons of tailings @ 5 per cent. Pb to tail-race.

No. 16-B.—Six spigots on each section (24 in all), with $\frac{5}{16}$-in. openings and 27-in. head, discharge as in No. 16-A to 3 distributing-barrels, feeding 50 tons @ 1/6.7 dilution to

No. 16-B-1.—Fourteen 6-ft. C. & W. frue vanners, same adjustments as above, making 4 tons of concentrates @ 45 per cent. Pb and 46 tons of tailings @ 5.70 per cent. Pb.

A summary of the above products is given in Table I., and a screen-analysis in Table II.

It will be seen that in this system there is a gradual reduction of the size of the particles of middlings by means of rolls as far as they are efficient, and, finally, by fine-grinding ma-

TABLE I.—*Summary of Products.*

Machine Number.	Feed.	First-Class Concts.	Lead.	Lead.	Second-Class Concts.	Lead.	Lead.	Middl'gs.	Lead.	Tailings.	Lead.	Lead.
	Tons.	Tons.	Per Cent.	Tons.	Tons.	Per Cent.	Tons.	Tons.	Per Cent.	Tons.	Per Cent.	Tons.
1-A-Jigs	260	.....	.....	.....	25.0	53.0	13.250	80	9.8	155	0.40	0.620
4-A-Jigs	200	.....	.....	.....	2.3	40.0	0.900	160	5.6	35	2.30	0.805
6-A-Jigs	90	.....	.....	.....	10.7	53.0	5.671	30	5.3	50	1.00	0.500
6-B-1 Jigs	50	2.0	70.0	1.400	1.5	45.0	0.675	20	6.8	30	0.90	0.270
8-A-Jigs	50	4.0	79.0	3.160	1.0	60.0	0.600	14	5.5	30	1 70	0.510
4-A-1 Jigs	80	.....	.....	.....	1.2	45.0	0.540	65	4.7	14	2.40	0.336
4-B-4 Jigs	80	2.4	75.0	1.800	0.9	60.0	0.540	50	4.3	27	1.90	0.513
14-A-Tables	85	4.0	78.0	3.120	5.0	50.0	2.500	30	.....	43	2.30	0.989
14-B-Tables	30	1.0	78.0	0.780	1.0	60.0	0.600	8	.....	17	1.30	0.221
16-A-1, 16-B-1 Vanners	90	.....	.....	.....	8.0	48.0	3.840	.....	.....	82	5.40	4.428
14-B-1 Table	8	.....	.....	.....	0.1	38 0	0.038	.....	.....	8	2.90	0.237
Totals	.....	13.4	.....	10.260	56.7	.....	29.154	.....	.....	491	.....	9.429
Averages	.....	.....	76.5	.....	.....	51.4	.....	.....	.....	.....	1.92	.....

Total—all concentrates—70 tons @ 56.2 per cent. lead,

TABLE II.—*Screen-Analysis of Products.*

On or Through	No. 1. Feed Rolls No. 2.	No. 2. Discharge Rolls No. 2.	No. 3. Feed Rolls No. 5.	No. 4. Discharge Rolls No. 5.	No. 5. Feed Rolls No. 7.	No. 6. Discharge Rolls No. 7.	No. 7. Feed Chilean Mill.	No. 8. Discharge Chilean Mill.	No. 9. First Spigot Calumet Class'r.	No. 10. Second Spigot Calumet Class'r.	No. 11. Third Spigot Calumet Class'r.	No. 12. Fourth Spigot Calumet Class'r.	No. 13. Feed Hardinge Mill.	No. 14. Discharge Hardinge Mill.	No. 15. Overflow Esperanza Class'r.	No. 16. Overflow Calumet Class'r.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
On 15 mm	57.9	2.8	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
On 7 mm	28.4	47.5	44.2	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
On 3 mm	13.0	33.3	50.4	64.8	70.0	23.1	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Through 3 mm	0.7	.....	5.4	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
On 20-mesh	.....	7.7	.....	23.7	30.0	56.4	70.5	.....	.....	.....	.....	.....	0.9	.....	.....	.....
Through 20-mesh	.....	8.7	.....	5.5	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
On 40-mesh	.....	.....	.....	6.0	.....	9.6	24.5	9.9	33.9	6.8	0.4	.....	9.5	.....	.....	.....
Through 40-mesh	.....	.....	.....	.....	.....	10.9	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
On 60-mesh	.....	.....	.....	.....	.....	.....	2.6	9.9	38.1	23.6	6.5	.....	20.1	2.6	.....	.....
On 80 mesh	.....	.....	.....	.....	.....	.....	1.0	8.5	17.0	25.1	14.5	5.0	20.1	7.1	.....	.....
On 100-mesh	.....	.....	.....	.....	.....	.....	1.4	5.9	7.1	23.6	28.3	13.0	20.7	13.4	.....	.....
On 150-mesh	.....	.....	.....	.....	.....	.....	.....	6.5	2.3	11.2	23.2	27.0	16.3	16.8	.....	.....
Through 150-mesh	.....	.....	.....	.....	.....	.....	.....	59 3	.....	.....	.....	.....	12.4	60.1	.....	.....
On 200-mesh	.....	.....	.....	.....	.....	.....	.....	.....	1.1	5.3	15.2	32.0	.....	.....	2.98	2.71
Through 200-mesh (sand)	.....	.....	.....	.....	.....	.....	.....	.....	0.5	4.4	11.9	23.0	.....	.....	8.95	19.66
Through 200-mesh (slime)	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	88.07	77.63
Total	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.00	100.00

chines. At each step, the freed galena is taken off and the tailings produced by cracking the particles are disposed of before the material is again subjected to reduction. While this complicates the process and makes a large tonnage of middlings to be rehandled, it does not result in an excess of slimes, as the material is extremely hard. The middlings are not mixed with the original feed until it enters the slime department, and the jigs on the original feed have a uniform load, and are kept up to a high standard of efficiency.

Another object striven for is to keep the amount of circulating-water as low as possible. To do this, the grades of the launders have been made ample to carry the products with a minimum of water. The fine trommels are washed with pipe-sprays. The fine streams from these impinge on the ore as it rolls in the trommels and, having a constant head, deliver a constant volume of water into the system. The feed to the rolls and grinding-machines is first de-watered, and the wash-water necessary to carry the discharge is applied through spigots with constant heads from the de-watering-boxes. The total water extracted from the slimes amounts to 372 gal. per min. for from 550 to 600 tons of mill-feed.

The type of jigs used is the same as described by Caetani in the article first mentioned, except that they are all four-compartment, have a larger opening between the plunger-compartment and the hutch, take their wash-water through pipes under the sieves, and do not de-slime their feed.

Some of the newer types of machines, as far as the Cœur d'Alène is concerned, adopted in the mill, are the Chilean mill, Hardinge mill, Esperanza classifier, and Blake-Dennison conveyor-weigher. Two Cooper-Hewitt mercury-arc lamps have been installed on the jig-floor, and give remarkably distinct color-values to the quartz, quartzite, siderite, pyrite, blende, and galena of which the ore is composed.

Of the smaller details, the use of manganese-steel roll-shells "trued up" with carborundum bricks; revolving trommel roll-feeders; the drilling of holes in the bottom of the fine-elevator buckets to destroy suction; the use of pure rubber washers between the elevator-buckets and the belt, to pass the fine sand, instead of allowing it to lodge and wear the belt; the air-lift for returning the overflow-water from the ore-bins,

etc., for table wash-water; the complete installation, on the bearings in the plant, of Keystone grease-cups with copper pins; and the use of double-edged or overflow-boxes in the thickening-tanks, are more or less novel in Cœur d'Alène practice. These overflow-boxes, while simple, are remarkably efficient, and give a perfectly clear overflow.

All the tailings are automatically sampled by means of water-tipples which carry a "splitter" at regular intervals through each of the vertically-falling tailings-streams. Regular tonnage-tests are made on these tailings. The concentrates-tonnages are kept by recording the dry weight of each bin as it is loaded out. Each bin is also assayed, in addition to the smelter-lots. In this way a complete analysis of the losses and recovery in each department is made at regular intervals, and the lead contained in the tailings plus that in the concentrates is compared with the contents of the feed.

A daily record of the lead-tonnage introduced into the mill as well as that recovered from it, together with the cost of operation in labor and supplies, the running-time, etc., is kept and carried forward on a form in such a way that a complete report of operations up to any required point is available at any time during the month.

The Macquisten-Tube Flotation Process.

BY O. B. HOFSTRAND, WALLACE, IDAHO.

(Presented at a meeting of the Spokane Local Section of the Institute, Wallace, Idaho, May, 1912.)

SOLID bodies will float in a liquid by reason of lower specific gravity, or of peculiar form, or of repellent action towards the liquid which prevents them from overcoming the resistance of its surface-tension. To the last of these causes the flotation of ores is to be ascribed. Nearly all ores and gangues are specifically heavier than water, yet they do not behave alike in this respect, and they may be divided into two classes, those of floatable and non-floatable minerals respectively.

Upon a close examination of these two groups of minerals we find that those particles that will float on the surface of

water are close grained, have a metallic, resinous, waxy or shining luster, and are water-repellent, while the particles which sink are usually porous, have a dull, earthy, non-metallic luster, and are water-absorbent. It is also further observed that the floatable particles are usually sulphides, while the non-floatable particles are usually oxides, carbonates, and gangues. By taking advantage of these facts separation can be successfully made between these classes without respect to the specific gravity of either.

In order to secure the flotation of suitable particles, they must be properly introduced to the surface of the water, so as not to break its surface-tension. Important factors are, the size of the particles, the angle of introduction to the surface, which varies with the size of the particles, the velocity of the particles, and an intermittent application of air to the particles.

After several years spent in investigations, the Macquisten tube-concentrator, embodying these factors, was developed and perfected by A. P. S. Macquisten in Glasgow, Scotland.

The tube, which is the principal part of this machine, is a thin, cylindrical shell, $\frac{3}{16}$ in. thick, 6 ft. long, and 8.5 in. in outside diameter. Internally, the tube is cast with helical corrugations (screw threads), preferably of quadruple pitch, advancing 3 in. with every turn around the tube, the pitch between corrugations being 0.75 in. The form of the corrugations, which has been determined by much study and labor, plays a very important part in producing flotation.

The tube rests horizontally on babbitted bearings at each end. The bearing at the feed-end is a conical-shaped hollow bearing, through which the feed is introduced. The discharge-end of the tube connects with a cast-iron sand-box composed of a sand- and slime-hopper, the joint between the sand-box and the tube being water-tight, while the tube is free to revolve. At the end of the sand-box, directly opposite the discharge-end of the tube, an opening is cut for the overflow of the surface-water, carrying the floating material. At this point a weir regulates the depth of the water in the tube. The sand-box is equipped with an adjustable siphon on the slime-hopper, by which the amount of water overflowing the weir is regulated. The tube is equipped with a machine-cut cast-iron spiral gear at the feed-end, with teeth cut at an angle of 30° to its center-

line, and is driven by a spiral pinion of steel with teeth cut at an angle of 60° to its center-line. The pinion is mounted on a steel shaft and operates at right angles to the tube.

The tubes are ordinarily erected in sets of eight, in double series of four, and are supported on a steel channel-frame, and so arranged that the two tubes in the same horizontal elevation are driven by the same shaft. Under each sand-box is a secondary feed-spout which feeds the tailings from the upper tube to the tube below it; the top tube only being equipped with an initial feed-spout, which receives the feed as it comes from the feed-distributors.

The operation of the tubes is very simple; the feed is received through the initial feed-spout into the first tube; is moved forward by the aid of the internal corrugations to the discharge-end of the tube; and in its advance is repeatedly carried from under the water, where it rolls and slides back, through and on to the surface of the water, and the particles remaining on the surface are carried over the discharge-weir in a thin film by the surface-flow. The minerals which adhere to the tube, and are carried over to the opposite side, are introduced to the surface at the same angle and are floated in the same manner. The tubes are operated most successfully at a speed of 30 rev. per min., and in tubes that are cast with quadruple thread the feed remains in the tube for a period of 48 sec. before it is finally discharged. At this point the tailings can either be sent to the creek (depending on the grade of feed), or can be sent to another tube or tubes in series, as may be required.

The capacity of the tube is not controlled by the amount of feed that can be put through, but is entirely dependent on the carrying-capacity of the surface of the water and the speed of the surface-flow. An average speed of the surface-flow is about 10 ft. per minute.

The amount of mineral that can be commercially conveyed in a thin film through the tube during a given period varies with the specific gravity of the mineral and the fineness to which the ore is crushed, as the film of mineral which will float off must contain one layer only. The actual amount of zinc-concentrates that can be floated during a period of 24 hr. is from 175 to 200 lb., containing about 48 per cent. of Zn. In

lead-flotation, the amount of concentrates, by weight, would be considerably greater.

The range of size of the particles amenable to flotation is from 20-mesh (hole $\frac{1}{40}$ in. sq.) down to the impalpable slimes. The amount of impalpable slimes is usually from 45 to 50 per cent. of the material passing through a 200-mesh screen (hole 0.003 in. sq.).

The extraction of which the tubes are capable, depends on the character of the ore, the fineness to which the ore has to be ground to free the mineral, etc. The efficiency ranges, by actual tests, from 50 up to 85 per cent.

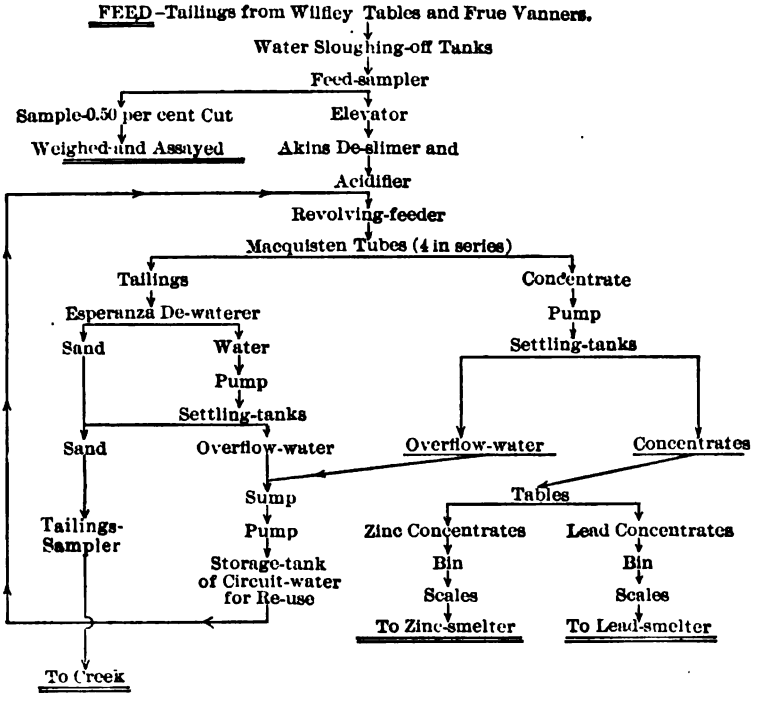

FIG. 1.—FLOW-SHEET OF THE MACQUISTEN-TUBE PLANT AT THE MORNING MINE, FEDERAL MINING & SMELTING CO., MULLAN, IDAHO.

In preparing the feed for the tubes (see flow-sheet, Fig. 1), it is first de-slimed; that is, all the impalpable slimes are removed, leaving nothing but granular material as tube-feed. The principal reason is not that the fine slimes cannot be floated, but that they are difficult to settle, since both mineral

and gangue in this finely-divided state will all float, not only on the surface but under the surface as well, remaining in suspension in the water. This tends to produce a low-grade, sometimes unmarketable, shipping-product. A second reason is the enormous surface-area which a small amount of the finely-divided sulphides will occupy, floating as they do, like the coarser material, only in a thin film. As an illustration, let us take a 30-mesh particle, which is about $\frac{1}{60}$ in. in diameter, and occupies a corresponding area. If this 30-mesh particle were crushed to pass a 500-mesh screen (were such a screen to be had), we would have particles of about 0.001 in. in size, and about 4,630 pieces, which would require approximately 17 times as much surface as one 30-mesh particle, disregarding the space between the particles.

The feed, after being de-slimed, is passed through a dilute sulphuric-acid bath, of from 0.05 to 0.5 per cent. strength, which has a tendency to clean the particles and also to assist in rendering the gangue particles resistant to flotation. After the feed has reached the tubes, a small quantity of a dilute solution of soap and oil is added at the feed-end of each tube for the purpose of increasing the surface-flow carrying the mineral particles through the tube.

The cost of treatment, based on a capacity of 100 tons per 24 hr., including cost of acid, oil, power, attendance, repairs and supplies, is approximately 40 cents per ton, using four tubes in series. The cost per ton based on 200 tons capacity, using four tubes in series, will not exceed 30 cents per ton of ore treated. The actual cost for sulphuric acid, soap, and oil used in the solution at the Morning plant is less than 1 cent per ton of ore treated.

The first installation of tubes was made at the Adelaide mine of the Glasgow & Western Exploration Co., at Golconda, Nev. The separation made was between chalcopyrite, rather finely disseminated, and a quartzose gangue, containing both spinel and garnet.

The second, and most important, installation was made at the Morning mine of the Federal Mining & Smelting Co., at Mullan, Idaho. The material treated is Wilfley-table tailings, and the separation made is between sulphides of zinc and lead, and siderite and quartz gangue.

A third installation of tubes is at Park City, Utah, where the material treated is a Wilfley-table middlings, separating zinc and lead sulphides from iron pyrites, silica, and lime.

The tube-concentrator combines simplicity in construction, compactness, and accessibility, and requires a minimum of power to operate. It is owned and manufactured by the American Direct Concentrating Co., of Salt Lake City, Utah.

The San Poil Mill, Republic, Wash.

BY EDWARD C. MORSE, REPUBLIC, WASH.

(Presented at a meeting of the Spokane Local Section of the Institute, Republic, Wash., July, 1912.)

THE metallurgical history of Republic has been completely reviewed in a paper presented at a previous meeting of the Spokane Local Section by Professor Thomson;[1] therefore, the present paper will be confined to matters directly pertaining to the San Poil mill.

When I came to Republic last fall to look over the situation there was a plant in operation successfully treating the tailings from one of the earlier mills in the camp. I refer to the mill of the Rathfon Reduction Co. This mill was operating in accordance with modern cyaniding practice, and I was given to understand that the work was very satisfactory and profitable, else it would not have been continued as long as it was. At that time I obtained much metallurgical information of value from Harry W. Newton, Superintendent of the mill of the North Washington Power & Reduction Co.

I took a number of samples from the San Poil mine, with the view of determining if Mr. Newton's experience with the ores from the other mines applied to the San Poil ores. I made a large number of tests under a variety of conditions, and found that it was possible to extract as high as 98 per cent. of the values from San Poil ore by the laboratory-tests when the ore was pulverized to 200-mesh and treated with a solution of a strength of 5 lb. of KCN to the ton. A fairly strong solution was required because of the proportion of silver in the ore, which makes it, from a metallurgical point of view, practically

[1] Ore-Treatment at Republic, Wash., pp. 672 to 685, this volume.

a silver-ore. At present the strength of solution in No. 1 or strong solution storage-tank is kept at 7 lb. of KCN to the ton of solution. The weaker solutions are of a strength of from 3 to 4 lb. per ton. About 0.25 lb. of lead acetate is also used to the ton of ore. The consumption of lime is about 3 lb. to the ton of ore.

In designing the San Poil mill, there were several conditions that had to be considered. In the first place, it had to be confined within a plot less than 150 ft. square, between the Great Northern railroad and the wagon-road, and be kept away from the boundaries of certain mining-claims. Moreover, we were limited in the scale of the undertaking by the amount of money available. We have, however, succeeded in getting the mill completed, as well as a spur from the Great Northern railroad, delivering ore and fuel over the top of the ore-bins and coal-bunkers, a tramway leading from the mine of the company to the ore-bin at the top of the mill, water-supply arrangements, a good substantial building with modern milling-machinery and tank-equipment, all resting on bed-rock with concrete foundations, piping, etc., for approximately $50,000.

The ore-bins are of about 600 tons capacity, so arranged that the ore can be delivered to them either by tramway or by bottom-dump cars from the spur of the Great Northern railroad. This bin is partitioned into three compartments, one for the mine-run ore from the San Poil mine, one for the custom ore (at present used by the Knob Hill mine), and one for crushed ore which has been through the Williams hammer-mill.

The flow-sheet is, briefly, as follows: The ore is delivered by automatic feeders of the shaking type to a belt-conveyer which runs parallel to the bins and which conveys it to the Williams hammer-mill, which pulverizes mine-run ore to a fineness of $\frac{3}{16}$ in. in diameter.

This Williams hammer-mill is the most novel feature of the plant. It is, to a large extent, a copy of the Quenner machine, which has been used so successfully in pulverizing cement-gravel for placer-work in Mexico. The first machine of this type to be used for rock-crushing purposes is that operated by the United States government for making sand for concrete on the Celilo locks on the Columbia river, and this machine in the San Poil mill is the first one to be used in crushing ore

for milling purposes, though there are now many of them in use for treating sand for concrete and paving uses. It is a simple machine of large capacity, and if we can judge from the history of the Quenner machine and the machine at Celilo, the cost of up-keep is low. The consumption of iron on the beaters or hammers per ton of material crushed during the past three months has averaged about 1 lb. We are crushing finer than has hitherto been attempted with a machine of this type. The machine consists of a central shaft, running at a speed of 600 rev. per min. inside of a trommel built of grizzly-bars 2 in. deep, each with a 1-in. face and spaced $\frac{3}{16}$ in. apart, so that all material must be pulverized fine enough to pass a $\frac{3}{16}$-in. slot before it can leave the machine. On the central shaft are thirty-two 16-lb. hammers suspended by chains, making, of course, 600 rev. per min.; so, assuming that each hammer strikes one blow per revolution, there would be a total of 19,200 blows per minute from all hammers. When this fact is taken into consideration, it is not difficult to understand why it has been possible for periods extending over 4 or 5 hr. to maintain a crushing-capacity of 15 tons an hour of mine-run ore, crushed to $\frac{3}{16}$-in. size.

If the ore is fed in a slightly moist condition, there is a great deal more fine crushing accomplished than if it is fed to the machine in a dry state, but the dry crushing develops a much larger capacity.

Sizing-tests made on the product of the machine have shown as high as 40 per cent. of the material (hard Republic rock) passing 20-mesh, of which 65 per cent. was finer than 10-mesh, and 6 per cent. as fine as 200-mesh.

When the material is fed dry, there is approximately about one-half as much fine crushing accomplished as when fed in a damp condition. This machine has a capacity several times that of the rest of the mill, so a portion of the bin already referred to has been partitioned off to serve as a storage-bin. The crushed material from the trommel or cage of the mill drops on to a belt-conveyor which delivers it into a bucket-elevator to be dumped into the storage-bin. On the way to the storage-bin, it is automatically sampled by a series of splits of my own design, and the sample is conveyed by gravity down a chute to the sampling-rooms, which are equipped with Jones samplers, chipmonk crushers, and a disk pulverizer.

The material from the storage-bin passes through a rolling feeder, designed by F. W. Bewley, General Superintendent, to a set of Allis-Chalmers, style "C," finishing-rolls, which are provided with carborundum blocks to overcome the bad effects of grooving and pitting. The rolls crush the material from the hammer-mill to 8- or 10-mesh size; from them it passes in a launder to a Gates tube-mill 5 by 22 ft. The tube-mill product is elevated by a tailings-wheel of our own design to a duplex Dorr classifier. The overflowing slimes pass from the Dorr classifier to a Dorr thickener. The sands from the Dorr classifier are returned to the tube-mill for regrinding. From the Dorr thickener, the thickened product in the bottom passes through 9 cone-bottomed air-lift agitators 12 ft. in diameter by about 18 ft. high, the tanks closely resembling the Grass Valley form of the Pachuca tank, but having a piping and circulating arrangement of our own design, which is an effort to escape the troubles usually encountered in standard practice. These tanks are connected in series, and the material passes from one to the other by overflow. The material from No. 9 agitator is delivered into a cone-thickener provided with an air-lift for loosening purposes, and from the bottom of it is delivered to an Oliver filter, which separates the solution from the pulp, washing the latter and sending it to waste, and returning the solution to four storage-tanks provided with sand-filters, to be used again. These sand-filter tanks have been changed to a series of natural-settling tanks. The solution in the storage-tanks passes through 24 iron zinc-boxes. The zinc shavings are packed in them in hanks or layers of rolls, each succeeding layer being across the one below it, to prevent by-passing of the solution, after the manner developed at the Butters Copela Syndicate mill. After the solution has gone through these boxes, and the gold and silver have been precipitated, it flows by gravity into the No. 9 agitator, where it joins the pulp which has been going for 30 or 40 hr. through the agitators, and goes with it to the cone-thickener. This gives it a preliminary washing on its way to the filter; the pulp being discharged from the bottom of the tank to the filter, and the solution from the top of the thickener to a small storage-tank, which delivers it to centrifugal pumps to be returned to the battery-tanks at the top of the mill, to be used again.

Under specially favorable conditions an extraction of from 93 to 94 per cent. has been accomplished, and it is generally better than 90 per cent.

We have a very good steam-power plant consisting of two 110 h-p. Titusville boilers, a 19 by 18 Erie automatic engine, a 10 by 16 by 10 Blaisdell low-pressure air-compressor for agitation, and a 60-kw., 220-volt generator for lights and for outside power.

The water is procured from a well in the gulch just below the mill, which is provided with a power-pump driven by a 2-h-p. motor. There is also a gravity water-supply from Mud lake.

The arrangements for supplying the plant with fuel are very convenient, but if, in the near future, cheap electric power would be supplied to the operators at about the same cost as in the Cœur d'Alène and other mining-districts, it would be a long step in advance for Republic, for it would probably reduce the cost of mining and milling $1.50 a ton and double the tonnage of available ore that could be mined and milled at a profit in Republic camp.

When this paper was originally read, the mill was just commencing operations. After several months' experience in operation, in which more than $30,000 in bullion had been produced, several important changes have been made. The hammer-mill, while coming up to expectations in crushing-capacity, was so light in construction as to have given considerable trouble from breakage of various parts, and the manufacturers of the machine are supplying a new mill of about double the weight of metal to enable it better to withstand the strain.

An intermediate pebble-mill, manufactured by the Traylor Engineering works, of Allentown, Pa., is to be installed in place of the light set of rolls at present in use between the hammer-mill and the table-mill, in order to increase the crushing-capacity. This pebble-mill is to be 8 ft. in diameter by 6 ft. long, with peripheral discharge, using extra-large pebbles. It is also intended to install an additional Dorr thickener between the No. 10 agitator and the Oliver filter, so as to give the ore a replacement-wash with barren solution before it goes to the filter, thus relieving the Oliver filter of half its work by

reducing the grade of the material fed to it. A stacker, or belt-conveyor, has recently been installed for removing the tailings from the filter, which is at present discharged dry. This conveyor, about 75 ft. long, is housed in to protect it from the cold weather.

The new hammer-mill is to have the grizzly-bars on the trommel spaced 0.25 in. apart instead of $\frac{3}{16}$ in., as used on the present machine. The purpose of this is to overcome the occasional difficulty due to wet ore clogging the trommel-screen.

While the cost per pound of hammers for the hammer-mill is quite high, it does so much crushing per horse-power that we feel that it can be well afforded.

Although it has been demonstrated that the hammer-mill is a very effective crushing-machine, it cannot be said to be past the experimental stage until the newer and heavier machine has been fully tested.

The mill has developed a capacity approaching 100 tons a day since installing ribs on each alternate liner in the tube-mill.

The output of bullion ranges from $600 per day to as high as $900 per day. If the same grade of ore holds out it is expected that the output will exceed $1,000 per day when the new machinery is installed and the crushing-capacity has been increased to an average of 125 tons daily.

The San Poil Co. has the following executive staff: President, Robert A. Koontz; General Manager, J. W. Turner; General Superintendent, F. W. Bewley; Mine-Foreman, William Wagner; Metallurgist, Edward C. Morse.

DISCUSSIONS.

Slime-Filtration.

Discussion of the paper of George J. Young, *Trans.*, xlii., 752 to 784.

ASKIN M. NICHOLAS, Melbourne, Australia (communication to the Secretary *): Professor Young deals with quite a variety of machines and methods, but, strangely enough, makes no mention of the work done by me, although he concludes with the remark that the "thin-cake continuous filters are a decided improvement over the thick-cake filters."

As those interested in slime-treatment may consider that I did nothing more than to patent a process, I submit Fig. 1, showing the working-drawings of a machine, made, tested, and proved to work satisfactorily by me early in 1905. This time is prior to any work done in slime-filtration with rotating filters by E. L. Oliver or others. I maintain that a number of appliances now in use for slime-filtration infringe my patents. And after most exhaustive inquiry, I know of no machine or process, prior to my description in 1899,[1] that embodied getting rid of the residues by holding them on the cloth with the vacuum, and then discharging them with back-pressure, so as to form a cycle of the operation, the result being continuous treatment. I therefore think that any paper dealing exhaustively with this subject is not complete without a reference to the work that I have done.

* Received Mar. 11, 1912.

[1] U. S. Patent No. 619,211, Feb. 7, 1899.

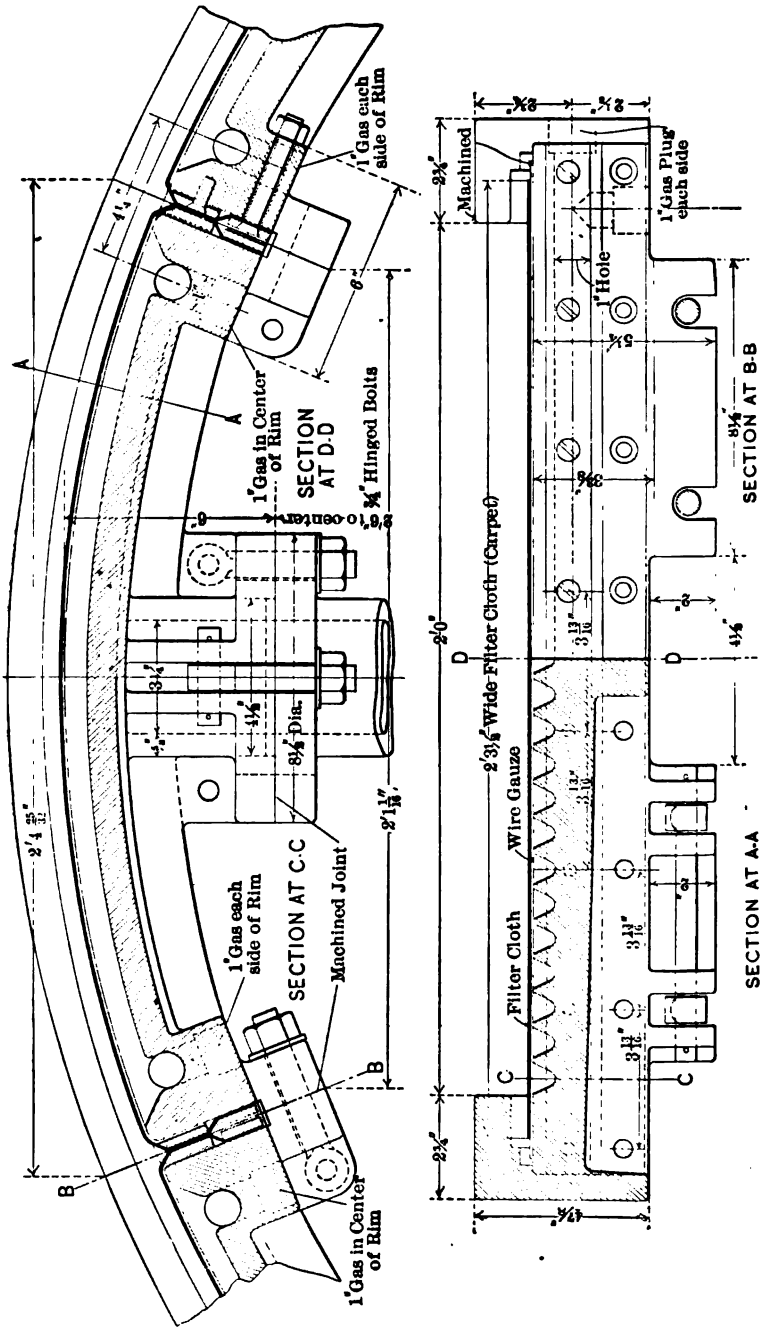

FIG. 1.—DETAILS OF FILTER-WHEEL.

Notes on the Laramie Tunnel.

Discussion of the paper of David W. Brunton, p. 99.

W. L. SAUNDERS, New York, N. Y. (communication to the Secretary *):—The Laramie tunnel, though a small one, compares very favorably in the speed of driving with the great Alpine tunnels which have the record so far. I think it safe to say that, taking all things into consideration, the record made at Laramie is the best American record, though it still falls short of the long-tunnel records abroad.

In discussing tunnel-records it must always be borne in mind that there are conditions peculiar to each individual tunnel job, so that no comparisons are complete, fair and conclusive.

At Laramie there is one point which stands out prominently, and which distinguishes this from any of the recently-built Alpine tunnels, and that is that here we have great progress in an American driven tunnel without the use of a carriage for mounting the drills. At the Arlberg and Loetschberg tunnels, the carriage was an essential condition for progress. All the heading-drills were mounted upon a bar which rested upon a small carriage. This carriage was withdrawn after the drilling and run into the heading after the blast without much disturbance of the drills, and with very little adjustment except that required in jacking the horizontal bar across the tunnel. The carriage was so small in its track-area that it required little mucking to run it in to the face; and as the rock-drills used were heavy machines 3⅝ in. in diameter, progress in setting up and drilling was very much accelerated by the use of the carriage.

At the Laramie tunnel hammer-drills were used instead of piston-drills, and here we have an important distinction which

* Received Mar. 23, 1912.

admits of the use of a drill of light weight, and if drills of light weight are to be used in tunnel-driving no carriage is necessary. At Laramie the same horizontal-bar system was used, but the Laramie bar was very much smaller in diameter and very much lighter than the Loetschberg bar because hammer-drills do not kick back like piston-drills. With this light bar and these light drills the men were able to climb over the muck, jack the bar across the tunnel and begin drilling at the top while the mucking was going on below. This would be impracticable if large piston-drills such as were used at the Loetschberg had been employed at Laramie. Hence we may reasonably reach the conclusion that if light-weight hammer-drills will clean up as much ground in a certain time as heavy-weight piston-drills, there is no longer any necessity for a tunnel-carriage, and the simplicity of the plant is made possible.

In both the Loetschberg and the Laramie tunnels the headings were about 10 ft. wide; at least, this was the width of the first Loetschberg heading. At Laramie the height was 8 ft. and at Loetschberg 6 ft. 6 in. This additional height at Laramie called for the drilling of more holes and the removal of about 25 per cent. additional material. While this is a handicap in some ways, yet it is partly compensated for by the greater space it gives in which to work.

It is striking to notice, on comparing these tunnels, that at Laramie the actual drilling-time was more than double that at Loetschberg. At Loetschberg the holes were drilled to a depth of only 4 ft., this being repeated for each 8-hr. shift, while at Laramie the holes were much deeper, and it is readily understood that deep holes cannot be drilled at the average speed of shallow holes.

In the Loetschberg tunnel there were drilled 13 by 2 = 26 holes by 4 ft., equals 104 ft., and the actual drilling-time was 2 hr., or 52 ft. of holes were drilled per hour. In the Laramie tunnel there were 23 by 7.5 ft. = 172 ft., but the drilling-time was 5 hr., so that the number of feet drilled per hour was 172 ÷ 5 = 34.5 ft., as compared with 52 ft.

There is a simple and self-evident explanation of the great discrepancy in these drilling-times. At the Loetschberg tunnel there were four drills in constant use, so that the record of feet per drill per shift was 104 ÷ 4 = 26 ft. in 2 hr., or 13 ft. per

hour. In the Laramie tunnel, although this item does not appear in the paper under discussion, only two drills were at work constantly, and a third drill less than half the time, 2.4 drills being above the average. Then $172 \div 5 = 34.5$ ft. per hour $\div 2.4 = 14.4$ ft. per hour per drill, which figures out a slightly higher drilling-speed at Laramie, even with the disadvantage of drilling the deeper holes.

If it were not that the other operations of the cycle were done with remarkable celerity the time-record would not have been nearly as good as it is for the Laramie tunnel, and it is evident that if more time could have been saved it would have been in the actual drilling, by using more drills.

Mr. Brunton states in his paper:

"European tunneling-methods were copied as closely as the American wage-scale and differences of conditions would permit."

In this, of course, he refers to methods only and not to plant, because there is a distinct difference in the two systems so far as plant is concerned, though there is little or no difference in the system of blasting employed.

An important feature about the plant at Laramie is the fact that the drills were used with air and water passing through the steel to the bottom of the hole. This is known as the Water-Leyner system of drilling, and it is likely that the progress made in this tunnel is to a large extent due to the introduction of water and air at the bottom of the hole. This not only discharges the cuttings as fast as they are made, but it produces a clear atmosphere in the heading, enabling the men to work better. Air and water when mixed at the bottom of a drill-hole should not be confused with air alone or water alone. To attempt to blow out the cuttings in a heading-hole by air alone is to create a dust atmosphere that in time will produce sure death to those living in it. Furthermore, it clouds the headings and interferes with good work. The introduction of water alone does not accelerate the drilling as much as does the air-discharge. It has the further objection of being more expensive, since it requires water under pressure to be led to the heading. It is also undesirable because it makes a wet and foggy atmosphere in the heading.

H. Foster Bain, San Francisco, Cal. (communication to the Secretary*):—Mr. Brunton's paper will be welcomed by all engineers interested in tunnel-work. The results achieved in driving the Laramie tunnel mark a distinct advance in methods, and it is interesting to note that the progress is in the direction of European practice, namely, shorter holes and more rounds. The problem in rapid tunnel-driving is well recognized to be that of handling broken material in constricted space, and in Europe the effort has been to solve this by breaking less rock at one time, but firing oftener. A few years ago, in the United States, the large percentage of time used in setting up the drills led to strong efforts to drill and shoot long holes. The new drills, however, have greater speed, and being lighter and simpler, are more quickly set at work. As a result it is easier to "get in a round" than to "get out the muck." By drilling shorter holes, loading them heavily, and using a horizontal bar, the time lost in shooting is greatly reduced. It is interesting to compare the work done in the Laramie tunnel with that at the Elizabeth tunnel of the Los Angeles Aqueduct. In April, 1910, the south portal was advanced 604 ft., W. C. Aston being in charge. During May the north portal was driven 567 ft. under direction of John Gray. These were the records for American work in hard rock, prior to January, 1911, in which month greater speed was made in the Laramie tunnel. Details regarding the work are quoted below, being taken from the *Sixth Annual Report of the Bureau of the Los Angeles Aqueduct*, of which William Mulholland is Chief Engineer, and J. B. Lippincott, Assistant Chief Engineer.

The Los Angeles Aqueduct crosses under the crest of the coast range 45 miles north of the city of Los Angeles. The Elizabeth lake is approximately 0.5 mile east of the center of the tunnel, and a number of small lakes are situated in the valley westerly from Elizabeth lake. The tunnel is 26,870 ft. or 5.09 miles long. The coast range has a double crest at this point, with the valley of the Elizabeth lake almost in the center of the tunnel-line. Where the tunnel crosses this valley, it is 250 ft. beneath the surface of the ground.

Work was started by hand at the south portal on Oct. 5,

* Received Mar. 23, 1912.

1907, and at the north portal on Nov. 1, 1907, and was so prosecuted until adequate machinery could be installed. It was supposed that this long tunnel would be the controlling factor in the time necessary for the completion of the aqueduct. Consequently, equipment of a substantial character was placed here, consisting of four 500-cu. ft. per minute, two-stage air-compressors, which permitted duplicate air-installation at both the north and south ends. One 18-in. positive blower, and a track of 36-lb. rails on which electric locomotives were operated, were furnished for each portal.

The tunnel was driven on a slope of 1 ft. in 1,000. The timbered section required a theoretical excavation of 5.02 cu. yd. per linear foot, and the untimbered section 4.18 cu. yd. This is to be a pressure-tunnel, and is the outlet from the bottom of the Fairmont reservoir. It will be a portion of the penstock of the first power-plant south from the crest of the range. The amount of water discharged through the tunnel will vary with the load on the power-plants up to 1,000 cu. ft. per second, and average 400 cu. ft. per second for the day. The fact that the tunnel diverts from the Fairmont reservoir will permit of this fluctuation. As the electric load-factor in Los Angeles is estimated at 40 per cent., the maximum flow of water from the tunnel is 2.5 times the mean. The hydraulic gradient, therefore, will vary with the volume discharged, and will be cared for by the depth of 80 ft. of water over the intake of the tunnel.

The ground along the north half of the tunnel was very much broken and consisted mostly of decomposed granite. Much of the ground was very heavy, requiring close timbering, and in numerous places the tunnel had to be retimbered two or three times. Large flows of water were encountered, and also swelling ground. The excavation of this portion of the tunnel was most difficult, aud called for courage, skill, and persistence. At 1,117 ft. from the north portal, a large fissure filled with sand and water was encountered which broke through the face of the tunnel, and could not be passed from that side. This caused considerable delay. Therefore, a shaft was put down 3,000 ft. from the portal, and the heading driven each way therefrom, in order to maintain the progress-schedule and to approach the dangerous ground more guardedly from

the south. This running water and sand were finally overcome by driving overlapping steel rails in advance of the heading, and closely following with careful excavation and timbering. John Gray, superintendent for the north end, has accomplished a remarkable work in the driving of this half of the tunnel without further mishap.

The south half of the tunnel was in a gneissoid granite, in some places rather soft, and in others hard. Broadly speaking, it was ideal tunnel-ground. This portion of the tunnel, as a rule, required no timbering. The south end was in charge of W. C. Aston, tunnel superintendent.

In driving the tunnel in hard rock, the full section was drilled and shot, each shift getting in a round and firing it. Where the ground was heavy at the north end, a lower heading was driven in advance, using false sets and crown-bars which carried the weight of the roof ahead of the front set. The tunnel was then widened by putting in the permanent posts, the temporary floor from which the upper section of the tunnel was excavated and mucked resting on them, the roof-segments being placed as the excavated material was removed, the whole process being one of a carefully detailed advance.

When this tunnel-work was started, it was estimated that a reasonable progress for each end would be 8 ft. per day with three 8-hr. shifts, and a bonus-schedule was adopted by the Board of Public Works, which provided that each man working in the tunnel, up to a shift of limited size, would receive a bonus of 40 cents for each foot that this schedule was exceeded, the progress to be measured at the end of each 10-day period. This bonus was paid in addition to the regular wages for tunnel-work in this section, the men receiving these wages whether the bonus-schedule was exceeded or not, and the bonus being distinctly a reward for extra exertion. As the costs for driving the tunnel are quite constant for each day's work, the cost per foot consequently would vary with the rate of progress, and it was estimated that this bonus would be considered as the men's equal share in the saving to the city resulting from the beating of the base-rate.

From the south portal, 13,500 ft. of tunnel was driven, and from the north portal, 13,370 ft. The average rate of progress at the south portal was 11.11 ft. per day; at the north portal,

after the connection was made through to the shaft, the average progress was 13.65 ft. per day. From the portal to the end of the section driven from the north end, including delays on account of shaft and cave-ins, the average progress was 11 ft. per day. The average progress for the two headings for the 1,215 days' work was 22.1 ft., or a little better than 11 ft. per day for each end. The connection of the two headings was made on Feb. 28, 1911, after the expiration of 40 months of work. (The time which the Board of Engineers estimated as necessary to complete this work was 5 years.) The center-line of the tunnel met within $1\frac{1}{8}$ in. and the grade checked within $\frac{5}{8}$ in. The total cost per foot for the driving of the tunnel, including administration, equipment, surveys, etc., but not lining, was $44.80, and the saving over the estimate of the Board of Engineers was about $500,000. An interesting feature of this work was that the tunnel was driven as rapidly and as cheaply near the center as it was near the portals; in other words, the transportation by electric locomotives and the strong ventilating-system were adequate to prevent delay. The organization of the work and the experience gained by the engineers steadily reduced the unit-cost as the work proceeded. While almost the same distance was driven from both portals, the work at the north end was interrupted by excessive flows of water, which had to be pumped out. It is noteworthy that at the south portal of the tunnel, one 18-in. blower was sufficient to ventilate it quickly up to the last 1,000 ft. Great credit is due to the superintendents at both ends of the tunnel for the records made, in both speed and economy.

A Concise Method of Showing Ore-Reserves.

Discussion of the paper of N. H. Emmons, p. 322.

E. W. KING, Bozeman, Mont.: The form of measuring up ore in sight looks very plausible, as illustrated in the paper of Mr. Emmons, but from my experience of many years of mining in Montana and Nevada, I have found it is very hard to measure up anything nearer than a guess of what you have in sight, except in the case of a true vein, or a flat vein where the boundaries are well defined. I have been interested in several mines with Dr. A. R. Ledoux, and I know that my foreman and myself, going into the mines and using our best judgment, could seldom see more than 30 days' supply actually in sight, and yet we were running from 100 to 250 tons a day, and keeping it up year after year.

PROF. JOHN D. IRVING, New Haven, Conn.: I understand from Mr. Emmons that this is a method which has been found applicable in the copper-mines in Tennessee, where the record could be made with a good deal of accuracy, and where the work in the mines allowed the necessary development to be done, so that the record could be kept up. I do not think there was any intention in his mind to imply that this was applicable to all cases, especially such as Mr. King has mentioned.

PRESIDENT CHARLES KIRCHHOFF, New York, N. Y.: I think the point is that the paper impresses the necessity for doing it where the possibilities exist. There has been too little attention given to this very matter, particularly in the United States; but I think our English friends are following it very much more zealously, and the demand for it there is possibly greater than it is here, because English boards of directors handle mines that are often very much further away than our own are, and the English public and the English engineers generally are very much more insistent upon seeing how much longer current dividends are going to last, or current assessments are going to continue.

WILLIAM KELLY, Vulcan, Mich.: My only criticism of Mr. Emmons's paper is that the adaptability of the method to various conditions is not quite as clearly stated as it might be. In the Lake Superior region, with which I am most familiar, a system such as this could probably not be used. In ore-bodies running from zero to 200 or 300 ft. in width, a vertical longitudinal section would not represent the whole width. The difficulty usually to be met is not so much in estimating the quality of the ore in the developed part of the mine as in estimating the ore in that part of the mine which is only partly developed, or in that part which is at present beyond development, and yet to which we have to give some value.

DR. N. S. KEITH, Philadelphia, Pa.: This manner of exploring a mine becomes most important in the low-grade ores, such as are found, for instance, in Arizona. At the Ray Consolidated mine they have, according to latest reports, about 30 miles of underground work, done simply for exploration, the reason being that such a large amount of capital is necessarily expended in the erection of mills and works, there must be a reasonable certainty of profit for a number of years, in order to return the capital invested. The little Ray Central mine, which is about to be absorbed by the leviathan, next to it, has about 6 miles of underground work, extending in depth to about 400 ft., and including levels, drifts, and winzes, etc., which have exposed bodies of workable ore of various sizes. These have been cut up into blocks about 200 ft. square at the various levels, and winzes have been run to determine the amount of ore available. The Ray Consolidated, though it has but just begun its concentration-works, has in this way developed more than 77,000,000 tons of ore. The Ray Central has developed something like 12,000,000 tons. One body, for instance, was found which the engineer estimates to contain 700,000 tons of ore, carrying more than 6 per cent. of copper.

There seems to be no other way to determine whether the necessary money should be spent upon the erection of works than to make these extensive developments beforehand. The Goldfield Consolidated people say they have developed two years' supply of ore, at nearly 1,000 tons per day. They do not think it was necessary to keep up greater development,

the geologist having reported that the ore will probably last for several years at least. At the Tonopah mine they keep ahead actually developed ore amounting to about two years' supply. They have pursued the same plan for several years, not doing any more exploratory work than seemed to be necessary, but keeping about two years of actual ore in sight.

At the Ray Consolidated, the maps are not kept on the plan proposed by Mr. Emmons, because their masses of ore are very thick blankets, and are divided into blocks of 200 ft. square, to represent the average values, by assaying the surface exposed, and doing so at every 5 ft. The Ray Central keeps all the necessary records at the mine, and has models of the mine on glass plates at the New York office. Their system of marking is somewhat different from that given by Mr. Emmons, and is very interesting and easily understood. The assembled glass plates represent the several levels; and the horizontal sections of the several ore-bodies at these levels are designated in colors. These, with the shafts, winzes, and levels, are easily seen in their relative positions.

Progress in Roll-Crushing.

Discussion of the paper of C. Q. Payne, p. 327.

E. G. Spilsbury, New York, N. Y.:—Do I understand correctly that the rolls are at first simple cylinders, one longer than the other, and that flanges are subsequently formed on the longer roll as the result of wear only?

Mr. Payne:—The flanges are formed by the wearing of the shorter roll into the other, which is about an inch longer. When the roll-shells are new, the longer one has machined at each end a ridge about $\frac{1}{8}$ in. or 0.25 in. higher than the face. These slight projections help to center and steady the rolls, by locating the flanges in advance.

Possibly they might be left off at the start; but Mr. Frazee has found it better, in machining the new roll-shells, to turn such small flanges upon the longer one. In his practice the feed-stream is extended a short distance beyond the two ends of the shorter roll-shell. This allows a small amount of crushing to be done between the flange-faces and the vertical ends of the shorter roll-shell, which prevents any ridges from forming at the ends of the shorter roll-shell.

In common practice, where two roll-shells of the same width are used, it is impossible to prevent ridging of the diagonally-opposite ends of the roll-shells. These ridges prevent accurate work in fine crushing. They also exert end-thrust upon the shaft, and strains upon the roll-frame. In Mr. Frazee's design the flanges not only act as movable cheek-plates, but they overcome all end-thrust upon the roll-shafts, and by preventing ridges from forming at the ends of the shorter roll-shell, they help to maintain the roll-faces in perfect parallelism while they undergo wear. The flanges become, therefore, of great importance in fine crushing. When they extend so far as to strike the draw-bolts of the shorter roll-centers, they are cut off.

PROF. ROBERT H. RICHARDS, Boston, Mass.:—One of the great difficulties in ordinary rolls is to get an even feed. How is the feed brought to this roll (indicating)? If the feed came more in one place than in another, that place would wear more.

MR. PAYNE:—That raises a very interesting question. Mr. Frazee has found that an even, or uniform, feed is not desirable. In fact, a uniform feed produces irregular wear. This is certainly true in the case of fine crushing, in spite of the general opinion of roll-manufacturers to the contrary.

The reaction of a feed-stream upon the crushing-surfaces of roll-shells is a subject to which little attention seems to have been directed heretofore. Mr. Frazee has found that in order to maintain the roll-faces parallel, it is essential to feed a larger amount of material along the sides of the faces than at the centers of the roll-shells, as explained in my paper. This seems to be partly due to the unequal mobility of different portions of the feed-stream as it is subjected to the crushing-pressure.

Differences of mobility would naturally cause differences of abrasion. A certain analogy may, perhaps, be found in the different rate of flow of metal in different portions of a hot ingot while passing between reducing-rolls. Whatever the reason may be, it has been found by actual practice that irregular abrasion of the roll-shells can be prevented by varying the thickness of the feed-stream in such a way that a larger amount of material is fed towards the ends than at the center of the roll-shells.

A further advantage of Mr. Frazee's roll-design, resulting from the high crushing-efficiency attained by control over the wear of the roll-shells and balanced end-thrust, is, that the roll-frame can be made considerably lighter than would otherwise be possible. By reason of the reduced strains, single side-frame castings, bolted together, can be used to support the roll-bearings in place of the usual massive continuous bed-plate casting. In actual operation these new rolls run almost as smoothly as a lathe. The 24-in. rolls are run at 100 rev. per min.; the 18-in. at 150, and the 12-in. at 250.

This system of crushing has been developed gradually. There are eight of these rolls now in actual operation, all exhibiting the same uniform control over the wear of the roll-shells. It is therefore not a "happy hit," but the result of close study of actual conditions on the part of the millman rather than of novel design on the part of the mechanical engineer. For this reason I think the new roll-design of Mr. Frazee's is of especial value and interest.

PROFESSOR RICHARDS:—There is a comic side to this: When we were preparing the first two volumes of our book on "Ore-Dressing" we had all our data in; and then the Wilfley table was developed, and that knocked the underpinning from under our plans. Now, we have our third and fourth volumes out, and we have taken great pains to show that flanges should be prevented, and now it turns out that the flanges are the best things to have. It reminds one of the old darkey's phrase, "The world sure do move."

GEORGE C. STONE, New York, N. Y. (communication to the Secretary *):—It seems to me that the principal novelty in the Frazee rolls is the feeder. Flanging one roll was common in Joplin a few years ago; while this helped to keep the roll-faces true, it was generally abandoned because the flanges, being made of cast-iron, frequently broke and caused much trouble when the pieces got into the jigs. Connecting the two adjusting-screws by a chain and sprockets is practically what has been done with other rolls. The feeder, however, is a new arrangement, and I believe a very good one. Even without the testi-

* Received Feb. 16, 1912.

mony of Mr. Payne's photographs, I should expect, from my own experience, that it would tend greatly to keep the faces true.

A few years ago we had a mill consisting of a 10 by 20 crusher, two 6 by 20 breakers in parallel, and a pair of 20 by 28 Krom rolls. When in good condition this just about kept the works in operation. We found, as Mr. Payne says, that the tendency of the rolls to wear hollow in the middle greatly reduced the capacity of the mill and necessitated much loss of time in shutting down to turn off the rolls. We first tried a pair of hard-steel rolls, but found that they greatly reduced the capacity of the mill, as the hard surfaces would not bite, but slipped, and the rolls packed full and threw off the belts, making it necessary to shut down and shovel out. Our next trial was with a feeder on the principle of Mr. Frazee's. While not as efficient as his, it decreased the trouble from uneven wear enough to answer our purpose. This device was a series of V-shaped riffles in the feed-chute. They were half round, about 0.5 in. in diameter, and placed with the points upward, so as to throw more feed to the sides than the center of the rolls. By a little experimenting we found what spacing and angle of these riffles gave us approximately even wear of the rolls. This very simple arrangement greatly increased the time the rolls could be run without requiring to be turned off, and enabled the work to be done without increasing the size of the mill.

S. Arthur Krom, Plainfield, N. J. (communication to the Secretary *):—Mr. Payne's statement that S. R. Krom was the "pioneer" in introducing into the United States crushing-rolls driven by belts, is, to the best of my knowledge, correct; but Mr. Payne's statement that S. R. Krom introduced high-speed rolls is not correct. As we understand the term "high-speed" —namely, a peripheral tire-velocity far greater than the falling velocity of the feed—such machines have never been built by any Krom manufacturer. Krom rolls have always been driven only slightly in excess of the feed-velocity. A high-speed roll, introduced a few years ago, naturally failed. Such a roll, revolving ahead of its feed, requires additional power, and is

* Received Mar. 27, 1912.

subject to additional wear on its bearings, without accomplishing any more work.

The ball-and-socket bearing, which Mr. Payne mentions, has only been adopted by those makers who were unable to design and build a roll with a "back-bone" stiff enough to stand up and do its work like a real roll. Such a bearing is a poor substitute, involving, as it does, extra parts in a machine that, owing to the nature of its work, should have as few parts as possible.

I think the use, for reducing large sizes, of the giant rolls mentioned by Mr. Payne, is, both mechanically and economically, a mistake. Rolls used for such work encroach upon the field of machines far better adapted to it. The cost of the jaw- and gyratory crushers for installation, power, and repairs is, in proportion to the work done, so much smaller than that of giant rolls as to counterbalance any advantage claimed for the latter in other respects.

The following extracts from a letter, dated Mar. 1, 1911, by a user of the giant rolls, who has had experience with jaw-crushers and three different makes of gyratory crushers, explain themselves:

"Our rock comes in very hard pieces; and at the time we made the change in our plant, a 42-in. opening was the largest gyratory crusher being built; so we decided to put in rolls. . . . .

"We found that in order to keep the rolls in fairly-good shape, we had to put on two new tire-plates each Sunday, at a labor-cost of from $30 to $50. Two plates with freight cost about $185, making a total of about $225 per week.

"It is one man's work to look after the oiling of the four bearings. If, for any reason, they should run dry for a small fraction of a minute, the rolls would be out of business. . . . .

"We operated the rolls 4 months and averaged 400 tons per day; but this small output cannot be charged against the rolls, as we had but one shovel and a great deal of trouble with the electric power.

"We had a 200-h-p. motor for each roll, and it took from 5 to 8 men with a lever to help start them.

"Three hundred and twenty 2.5- by 10-in. bolts are used to fasten the plates to the rolls. Occasionally a bolt breaks, and in going through the mill with the tailings is sure to do damage to the machines used for secondary crushing. In this way we broke three shafts in our No. 6 crusher.

"The price of 7 ft. by 3 ft. 6 in. rolls is about $25,000, in addition to a royalty of two cents per yard.

"If I were to build a plant now I would use a Kennedy machine. I believe his crusher, with the improved eccentric, is the best thing on the market to-day."

A jaw-crusher constructed upon certain lines, having a jaw-opening 10 in. by 20 in., and weighing not over 15,000 lb., will take a steady stream of 10-in. run-of-mine and reduce it without sliming to 1-in. size. To make such a reduction with rolls having equal capacity would require a roll at least 8 ft. in diameter, weighing about 80,000 lb. (see Fig. 1). The cost of the crusher would be about $1,500, and that of the roll, apart from the extra expense for foundations and transportation, about $8,000.

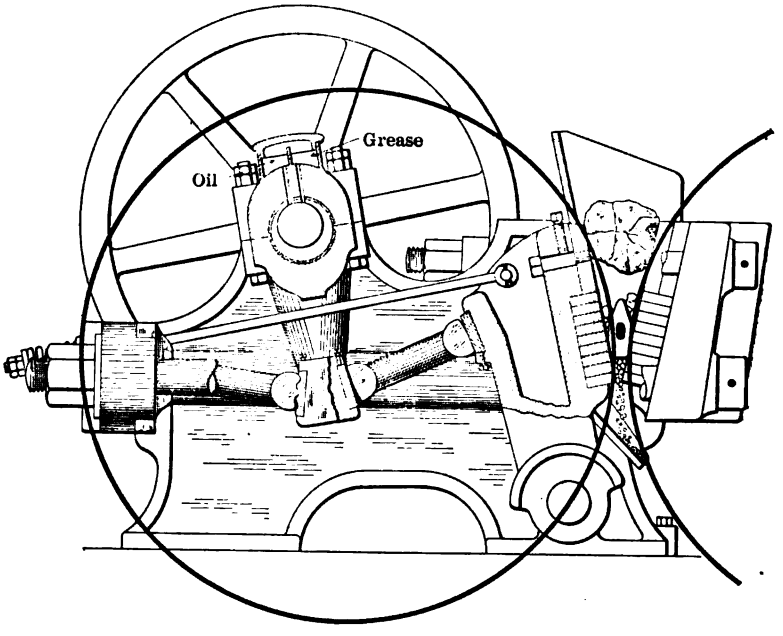

FIG. 1.—KROM CRUSHER.

Comparison of 10- by 20-in. Jaw-Crusher with Rolls for reducing from 10 to 1. The circular outlines represent Roll-Tires 8 ft. in diameter, which would be necessary to bite a 10-in. ore chunk.

The practice of dropping the feed from a considerable height to the roll, in order to drive it through by force of its falling-velocity, would permit the use of a somewhat smaller roll. But the height required is in most cases not available, hence additional expense for elevating would be involved. Moreover, the violence of this method of feeding and crushing would, in ore-milling, produce an excessive amount of slimes.

Again, it is desirable in such operations to remove the "fine

enough" material as fast as it is produced. The product of the giant rolls, containing a large amount of coarse material of various sizes, would require a very extensive screen-system to remove the fines, the ore passing through a number of graded coarse screens, until an oversize was obtained that would not destroy the screen removing the finished product. It is better practice to crush uniform at each step of the reduction, and screen near the end of the operation, thus making the screen-installation as simple as possible.

In ore-milling, the use of rolls is desirable when it is necessary to keep the slimes down to the smallest possible amount. The ability to do this is their most valuable feature, but this point has often been overlooked, and rolls have been used for purposes unsuited to their construction.

It has been found in general ore-milling practice that a reduction of 4 to 1 should be the limit for rolls, under the most favorable conditions. A reduction of 3 to 1 is the average, and often it should not exceed 2 to 1; so that in actual practice it would require a 30-in. roll working in connection with the 8-ft. roll to produce the 1-in. crusher-delivery size; the first roll reducing from 10 to 3, and the latter from 3 to 1.

A good roll, like a good horse, will stand (and generally receives) an awful lot of abuse; but experience shows that it does not pay to abuse a roll, any more than a horse. I once had a customer who insisted on reducing quartz from 2-in. to fine powder, with one set of 30-in. rolls. I could see that this man was anxious to be a "pioneer." Somehow or other he did not break the roll, but he did break his company.

Rolls for Fine-Crushing.—Mr. Payne says that "rolls for fine crushing, namely, to pass 30- to 40-mesh, etc., have not given satisfaction, and since little assurance can be given that they will do so and retain the granular feature of the crushed product, they have been assailed by various types of tube-mills, etc." My experience with rolls has taught me that the product of a roll is always granular, no matter how fine the crushing is. The attack on rolls for fine-crushing by tube-mills, etc., is to my mind justifiable, since, below 20-mesh, the capacity of the roll becomes too small for general mill-practice, in proportion to its installation- and operating-cost; and the capacity diminishes very rapidly with the fineness of the crushing.

Mr. Payne says that "the loss of capacity is due to the escape of the ore-particles through the rolls, owing to irregular or grooved tires." This is true to a certain extent; but the principal cause for the loss of capacity is the fact that the stream of mineral passing through the roll has, by reason of the fineness of its individual particles, become very thin, hence its tonnage drops off. Choke-feeding does not remedy this to any extent, and is difficult to regulate. The only remedy is a greatly increased surface for action, *i. e.*, more mineral surface to act on, and more crushing-surface to perform the work. Here the roll must give up the job to a machine fulfilling the above requirements.

Such a machine is found in the modern pebble-mill of the Hardinge type. The granular nature of the product from this machine is not lost to any damaging extent, and for fine-grinding operations the production of slimes can be kept within reasonable limits, considering the fineness of the work, by proper regulation of the feed and the adjustments of the mill. The mill is no more "fool-proof" than a roll is—which is no drawback in the ore-milling business. I have yet to see a milling-proposition that would pay a profit if operated by fools. I do not know where the term "fool-proof" machinery originated, but I think it must have been invented to fit some local condition in Joplin, Mo.

In regard to the grooving of tires, I have often found it due to uneven texture of the tire-metal. Under such a condition, grooving will take place, no matter how even the feed. A practical mill-man can easily arrange any one of several feeding-devices now on the market to deliver an even feed. In case of unavoidable tire-grooving, various methods of truing up the tires are successfully practiced. John Sargenson, of the Nipissing Reduction Works, Cobalt, has developed a simple apparatus for doing this while the roll is at work.

The Frazee method of dry-crushing, as outlined by Mr. Payne, could not be adapted to general mill-practice, since it would be impracticable or too expensive to dry the ore for crushing, and then wet it again for concentration, cyanidation, etc.

There is nothing new about the principal features of the Frazee roll-construction, such as the spring arrangement, ad-

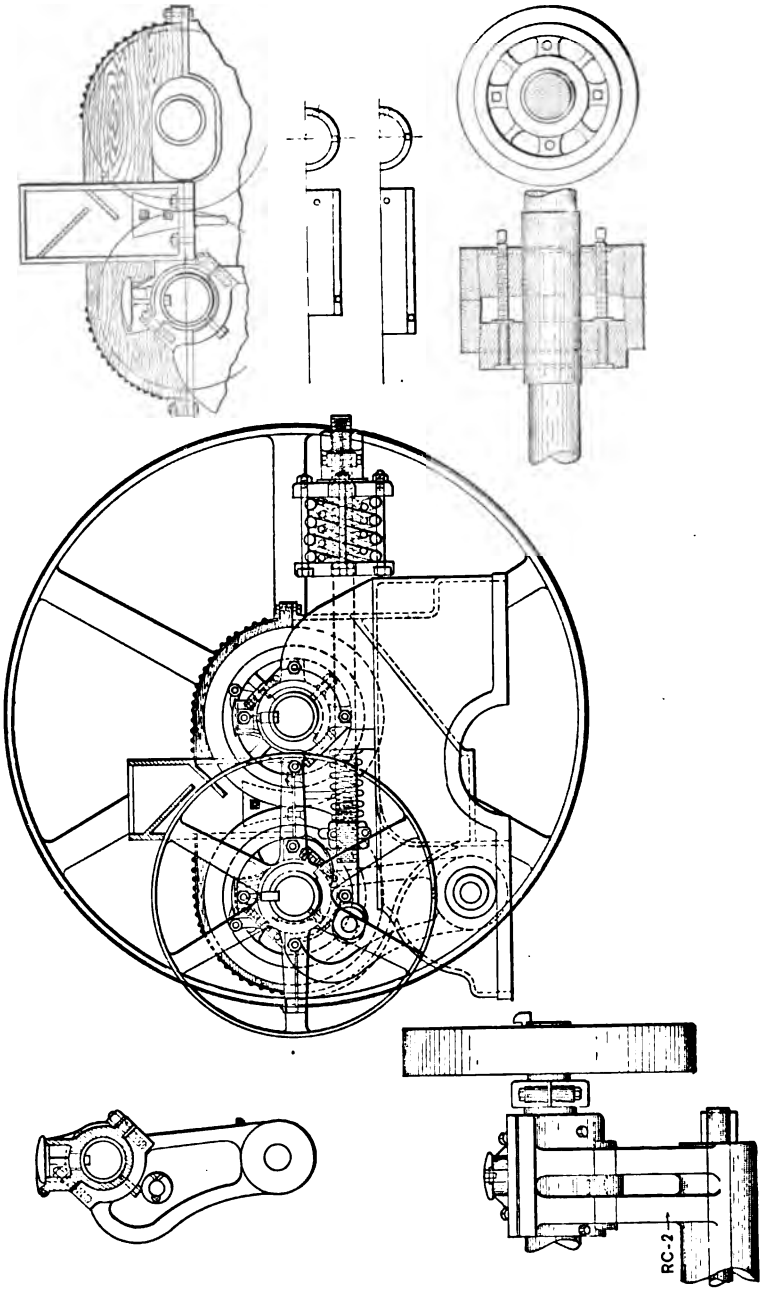

FIG. 2.—KROM ROLL.

Detailed drawing showing correct roll-construction. The movable bearings are connected by a solid yoke, RC-2. This yoke holds the movable bearings in perfect alignment and parallel to the fixed bearings.

justment, flanged tires, etc. They have all been used from time to time, and are now chiefly epidemic in the Joplin region, and are responsible, together with other local conditions, in making that district notorious for poor milling-practice.

The following illustration will further enforce and explain the foregoing remarks:

Fig. 2 shows detail-drawing of correct roll-construction. The movable bearings are connected by a solid yoke, RC-2 (see lower left-hand corner of drawing). This yoke holds the movable bearings in perfect alignment and parallel to the fixed bearings.

In the Hardinge pebble-mill,[1] the multitudinous crushing-surfaces formed by the contact of the pebbles with the walls of the mill, and between the pebbles themselves, is a mechanical feature with which a roll cannot compete. It will be noted from the results given in the accompanying test-card that with a comparatively coarse feed, 4-mesh, only 35 h-p. is required to give a capacity of 65 tons per 24 hr., reducing to sizes fine enough to meet the requirements of the average tailings-regrind.

CARD No. 33. DATE NOVEMBER 3, 1910.

Mesh-Test Card, Hardinge Conical Mill Co., New York.

Class of Ore :—Lake amygdaloid copper.
Mill used in test : 8 ft. in diameter, conical pebble mill.
The following shows mesh-sizes before and after reduction :

Through 4-Mesh.	Feed to Mill.	Product from Mill.
	Per Cent.	Per Cent.
On 8-mesh, ...	40.3	
On 10-mesh, ...	25.0	
On 20-mesh, ...	29.0	
On 40-mesh, ...	4.5	2.7
On 60-mesh, ...	0.5	8.5
On 80-mesh, ...	0.1	3.3
On 100-mesh, ...	.....	9.4
On 150-mesh, ...	.....	30.2
On 200-mesh, ...	.....	10.9
Through 200-mesh, ...	.....	34.0

REMARKS :—Capacity, 65 tons per 24 hr. ; charge pebbles, 8,000 lb. ; speed, rev. per min. of mill, 28 ; horse-power, 35 ; water, gal. per min., 15.

The effectiveness of the increase in crushing-surfaces is further demonstrated by the small percentage remaining on 40-mesh.

[1] *Trans.*, xxxix., 336 (1908).

In cases where it would be necessary to crush the original ore to 40-mesh or finer, before beginning concentration, it would also be better practice to carry the roll-crushing to about 6-mesh only, classify out the "fine enough," and finish the oversize with the pebble-mill. Slime-tables have now reached such a degree of perfection, both as to low cost of operation and high recovery of the values, that it would be much cheaper to install extra tables to take care of the slimes produced by the pebble-mill, crushing 6-mesh feed, than it would be to try to avoid making slimes, and at the same time crush very fine with non-sliming machines of a small capacity, making fine reductions.

C. Q. PAYNE, New York, N. Y. (communication to the Secretary*): I desire to comment briefly upon S. Arthur Krom's contribution to the discussion of my paper. In giving Stephen R. Krom the credit of introducing the belted high-speed crushing-roll into the United States, I may say that the difference in speed of the early geared Cornish rolls, which were sometimes driven as slowly as 4 or 5 rev. per min., and that of the belted rolls, which, when run at 100 rev. per min. and over, may be considered "high speed," as compared with the former, seemed to me sufficiently distinctive to be noted. The same distinction is made by Philip Argall in his interesting paper, Sampling and Dry Crushing in Colorado.[1] The narrow-tired roll, which was driven at an extremely high speed, was a later development, and proved to have other disadvantages.

I regret to notice that Mr. Krom makes a misleading use of quotation-marks in his comments by inclosing between them statements which do not quote accurately from my paper. I think, however, that this must be due to oversight or haste on his part, for I notice also that he entirely disregards Mr. Edison's remarkable use of rolls in pulverizing cement-rock, when he says that his experience has taught him that the product of a roll is always granular, no matter how fine the crushing is.

He has again overlooked my statement on p. 341, when he says that the Frazee rolls cannot be applied to general mill-practice owing to the necessity of drying the ore. No such

* Received June 25, 1912.

[1] *Transactions of the Institution of Mining and Metallurgy*, vol. x., p. 250 (1901–02).

necessity exists. I must also notice that while Mr. Krom classes the Frazee rolls with Joplin practice among things to be condemned, he finds the correct roll-construction in the Krom roll. As nothing is stated about feeding the Krom roll, this conclusion seems to be too general to be of value in this connection, or quite convincing, even if it might not have been inferred.

In regard to Joplin practice, the use of flanges on one of the roll-shells in some of the mills is certainly an interesting development. No feed-mechanism is, however, employed. The ore is carried to the rolls by a stream of water, and the object of the flanges is to direct the ore downward, and to prevent the water from splashing it out sideways while it is being crushed. These rolls are not used for fine crushing in the sense that Mr. Frazee's rolls are, and the flanges do not prevent grooving and irregular wear of the shells. The latter are made of cast-iron, and are given a very hard chill. When the chill is worn through they are discarded. As Mr. Stone points out, the distinctively novel feature of the Frazee roll-design is the method of feeding the rolls. Here the feed-stream is extended slightly beyond the ends of the shorter roll, so that by a certain amount of crushing between the ends of the roll and the flanges, the ridging of the ends of the shorter roll, as well as grooving, can be entirely overcome. The Frazee roll-shells can thus be brought quite close together, and in this way used for much finer crushing than the Joplin practice would permit. They have, therefore, little or nothing in common with the latter.

I am aware that it is not unusual for manufacturers of machinery to take a fling at Joplin practice. From a mining engineer's point of view, however, there is much to commend it. The Missouri zinc-deposits, for the most part, are irregular and uncertain in extent. Much of the ground has heretofore been leased to operators in comparatively small sections. The expense of simple mill-equipments can be more quickly recovered from operating-profits than more elaborate and expensive machinery. It is always "good practice" to make money, and rolls which would not be advisable for porphyry copper-ores, for example, under the peculiar conditions surrounding zinc-mining at Joplin have held their own, because they have justified themselves financially.

In my paper I have endeavored to press the point, that in treating those ores which must be kept in a granular condition while crushing them finely for concentration, the Frazee rolls hold out the promise of great usefulness by avoiding losses due to sliming. In Mr. Krom's comment I notice that he gives the screen-analysis of a material which has passed through a Hardinge tube-mill, but which does not have to be kept in a granular condition for concentration. The reference, therefore, does not seem pertinent to anything contained in my paper.

In regard to Mr. Krom's opinion on the subject of coarse crushing, his comparison between rolls and jaw-crushers on the basis of assumed cost seems decidedly unfair to the rolls.

I regret that Mr. Krom should rely for his opinion of Mr. Edison's giant rolls upon a correspondent who seems to have been unfortunate in not having secured the benefit of their enormous capacity, and who does not state the nature of the rock crushed. So indirect an attack may well be disregarded. Mr. Edison's mechanical genius does not need any tribute at my hands. His is the faith which literally removes mountains, and no one can watch the giant rolls in operation at Stewartsville, N. J., without increased respect for the human spirit which can measure itself with the enormous forces which the rolls unlock.

Whether the particular design developed thus far is found to be the best for the purpose or not, there is no doubt that all mining engineers will watch with keen interest the reduction of costs in mining and quarrying by means of coarse crushing, in which Mr. Edison has led the advance. He has certainly made a most notable contribution to the art of roll-crushing.

The Sintering of Fine Iron-Bearing Materials by the Dwight & Lloyd Process.

Discussion of the paper of B. G. Klugh, p. 364.

GEORGE W. MAYNARD, New York, N. Y.:—It is well known that there are many magnetite-mines of which the ore is too low-grade for direct use in the blast-furnace. For carrying out the sintering process, the preliminary step, in the case of magnetites, is the separation of the ore from the gangue by magnetic concentration. The resulting fine-grain product is very objectionable in the blast-furnace, and when used must be a very small percentage of the charge.

The sintering process has saved the situation by furnishing a product which is absolutely ideal on account of its cellular structure.

(In reply to questions by W. J. Tudor, and Joseph W. Richards, the following additional information was given by Dr. Maynard.)

The percentage of iron in the product is determined entirely by the degree of concentration of the raw material and is practically not increased by the sintering operation. The magnetic concentrates at Port Henry and at the Benson and one or two other mines contain from 65 to 67 per cent. of metallic iron.

The cost of sintering is from 39 to 41 cents per ton of product, including superintendence, depreciation, and repairs. The itemized figures are about as follows: Mixing (including labor), 24; power, 9; ignition, 5; repairs, 3; total, 41 cents.

PROF. J. W. RICHARDS, South Bethlehem, Pa.:—From a metallurgical standpoint I question the accuracy of the statement on p. 374: "*C.*—An increased reducibility of the iron so treated. (This has been conservatively estimated as 5 per cent. better than the best available ore.)"

It seems to me that when a material is sintered in this way, the constituents present, as lime, silica, and alumina, have a chance to form a compound with the iron oxide; that such a compound, if formed before the material goes into the blast-

furnace, will not be as easily reducible as the plain iron oxide itself, because the iron silicate, or whatever other combination of iron may have been formed during the sintering process, will have to be broken up; so that it does not strike me that the material can be as easily reducible as the unsintered material, or as the best available ore; the same objection, to my mind, would apply to the saving of carbon under the heading "*D*," in the formation of a slag. The formation of the slag from its constituents in the blast-furnace is a heat-producing or exothermic action, and if you produce the slag outside you lose the heat of formation of the slag from its constituents in the furnace.

In making Portland cement, the formation of the cement is an exothermic reaction which helps the sintering, and if the slag is formed outside the furnace, I should say it would rob the furnace of some of the heat of the formation of the slag. Of course, I admit the slag will be more uniform and the furnace will run more uniformly, but I do not think that there will be any heat saved to the furnace because the slag is formed outside. This latter comment, however, would not apply to the treatment of magnetic concentrates, since the slag-forming materials would be largely eliminated during the concentration.

Dr. F. W. C. Schniewind, New York, N. Y.:—As perhaps having a bearing on the question raised, Mr. Klugh says (p. 374): "This sinter now contains: the iron partly metallic and partly as either oxide." The question arises, how much is metallic iron, because the metallic iron would not require any reduction. It is already in a metallic state, and the additional heat which the reduction of the iron silicates will require in the blast-furnace may be offset by the saving of heat due to the metallic iron already produced.

Arthur S. Dwight, New York, N. Y.:—In connection with some of the tests which have been made of this iron-sinter, it has been definitely determined that in some cases at least a small amount of metallic iron is formed, probably as spongy iron, more or less irregularly mixed with the iron oxides, and probably some silicate when silica is present. As the

iron in the silicate would be in the form of FeO, I might raise the question whether the reducibility of the silicate of FeO would not be greater than when the iron was present as Fe_2O_3 or Fe_3O_4.

ANTON EILERS, Brooklyn, N. Y. :—It seems to me this could be easily settled. I believe in this sintering process FeO is formed, and as the material is at the same time very porous it takes a shorter time to reduce the FeO than it would Fe_2O_3. It is a fact that the sinter reduces very readily; at least, we find this to be the case in lead-smelting practice, where the same process is used for desulphurizing.

HENRY M. HOWE, Bedford Hills, N. Y. :—One of the speakers has said what I wanted to say, that part of the oxygen is taken out in sintering and less deoxidation is left to be done by the blast-furnace. Also, the mechanical condition of the ore is changed, so that the reduction can take place more rapidly with less fuel.

There is one statement which I think must be meant in a different way from what one would first take it. On p. 372 it is said that "taking off the ore and putting on the sinter has the same influence on the fuel-consumption as would result from taking off entirely that amount of ore-burden." That is to say, you would infer from this that the sinter is deoxidized without the consumption of any fuel, which, of course, is not what is meant.

MR. DWIGHT: The case referred to by Professor Howe was a replacement by sinter of a small portion of the ore-charge, and the effect of substituting this very open, porous, and readily reducible material was so beneficial that the fuel-charge present could be much more effectively utilized, and it was able to carry just that much additional burden. Naturally, we should expect to reach the limit of this possible increase of burden when the fuel came to be utilized to the best possible advantage.

In the case of lead- and copper-smelting, in which this sintering process is being rather extensively used, as mentioned by Mr. Eilers, it has been found that the fuel required for smelting can be lowered as much as 15 or 20 per cent., and yet give

equally good, if not better, metallurgical results, with a notable increase in the tonnage smelted by the same furnace.

To express the matter in another way, under ordinary lead-smelting conditions, the coke used would equal about 12 per cent. of the ore-charge, but when the charge contains a considerable proportion of sinter, this fuel-percentage can be reduced to 10 per cent. very easily, and sometimes very much under 10 per cent. The effect on the behavior of the furnace is quite marked, and can be readily accounted for by the unique physical properties which are peculiar to this product, among which may be mentioned the intimate mixture or propinquity of the elements to be smelted, and the porosity or cellular character, which gives the reducing-agents ample scope, and enables the blast to penetrate to every part of the charge.

Comparing the Dwight-Lloyd product with the product of older forms of pot-roasting, the former is found to have a very much greater reducibility; thus, in lead-smelting, the furnace which will show a speed of 100 tons a day of the ordinary pot-roasting product, will frequently smelt as much as 140 tons per day when the pot-product is replaced by Dwight-Lloyd sinter, as has been frequently demonstrated, all other conditions being identically the same, furnace, ores, fuel, and men.

With regard to the percentage of metallic iron present, there have been some analyses made of sintered iron-ore and iron flue-dust which show the relative proportions of FeO and Fe_2O_3, but I do not remember seeing any that have reported the metallic iron that was present as such. It has been observed, however, in certain cases where samples of the sintered product have been carefully broken up with a hammer, that small particles of metallic iron or iron sponge were present.

H. M. Chance, Philadelphia, Pa.:—May there not be a double action, first a reducing action, resulting in the formation of metallic iron, and as the carbon is burned out and the mass is cooling, may not a reoxidation of that metallic iron take place?

I ask if any investigation has ever been undertaken to determine whether this action does take place, whether there is a reduction followed by a reoxidation?

FELIX A. VOGEL, New York, N. Y.:—During this discussion nothing has been mentioned as to what forms the cementing action in this process. An inspection of the samples here exhibited will show that it is due to slag-formation, and not to any reduction of the iron-ore, either to metallic iron or to any form of oxide. It is merely a silicification forming a double or triple silicate of lime and iron. An examination of the sinter under a microscope will readily show small globules of glazed material which cover the ore. To my mind, this glassy cover will render the material less easily reducible than has been represented; in fact, it will take additional carbon in the blast-furnace to reduce this glaze, so subsequently to bring the iron oxide, or whatever oxidation it may be, in contact with the reducing carbon or reducing gases.

E. GYBBON SPILSBURY, New York, N. Y:—The samples of the sintered ore here exhibited do not appear to show any fines. Are any fines formed as the sinter is discharged from the grates, and if so, what is the percentage that must go back to the machine to be resintered, or else be charged to the furnace as fines, thus lessening the advantages which could be gained if it were all in lump form?

PROFESSOR MAYNARD: I have observed the work at Birdsboro on several occasions, and I have asked that very question which you have propounded. I am told that none has gone back for re-treatment. Of course, there is a certain dropping through the grate, a certain amount of breaking up, but I do not know that any has ever been carried back. During my observation on the few days that I have been there, I have not heard of that difficulty.

MR. DWIGHT: In order to determine the average size of the product resulting from treating flue-dust, I made a test on the second day the plant was started up at Birdsboro. On account of the large amount of carbon present in the flue-dust, which has to be burned out, there is a considerable amount of internal shrinkage, much greater than occurs when an ore like magnetite or hematite is used, where the exact percentage of fuel necessary to do the work can be added.

The product which results from the treatment of flue-dust

remains in the form of a solid cake until it is discharged from the machine, and then it tends to break up into crab-shaped masses or aggregations similar to the samples here shown. These aggregations are of various sizes, from pieces twice as large as the fist down to very small sizes. In order to determine the relative proportions I made the following experiment: the machine was stopped, the finished product carefully scraped off from several of the pallets, and the fragments classified into several sizes by a rough hand-sorting, throwing into one pile those pieces which were larger than a 2-in. ring, into another pile those that would go through a 2-in. ring, into another those which would go through a 1-in. ring, and also the pieces which would go through a 0.5-in. ring, and everything below that was screened through a 0.25-in. screen, and passed through a consecutive series of screens down to 120 mesh, and the various sizes were weighed and the percentages calculated. The proportion of each of the sizes coarser than 0.25 in. was from 20 to 30 per cent., and of the entire mass, nothing being lost, 98 per cent. remained on a 0.25-in. screen, and 2 per cent. passed through. Of that 2 per cent. which passed through the 0.25-in. screen, one-half remained on a 10-mesh screen, and one-half passed through.

When the process is conducted with a proper proportion of moisture, proper ignition, proper mixture and distribution on the machine, there should be practically no "fines" in the everyday practical sense. When irregularities occur, as they sometimes will, irregular spots will occur, and these will sometimes produce "fines." The amount that goes through the slot of the grate-bar is exceedingly small. In the course of a day there will accumulate, in the suction-box, perhaps a wheelbarrow full or two, which is cleaned out at the beginning of the shift and simply put back into the bins to be re-treated. It does not amount to enough to be considered serious. However, if on account of some peculiar condition an undue quantity of fines should be produced, it is very easy to keep it out of the final product by having the machine discharge on a grizzly which will deliver the coarse product into the cars, and the fines can be automatically returned to the machines.

There should be less fines made when sintering magnetites or iron concentrates than when flue-dust is being treated. The

carbon in the flue-dust is apt to run quite irregularly, so that frequent changes in adjustment may be necessary to maintain a perfect product. In the regular Dwight & Lloyd practice each fundamental requirement of the process is controlled by a separate mechanical attachment; so that the proportion of fuel in the charge, the permeability of the ore-bed, the volume and pressure of the gases, the degree of ignition, the percentage of moisture, the time of sintering, etc., can each be separately and immediately modified without disturbing any of the other adjustments. Thus, it is possible to keep the operation at all times up to the highest degree of efficiency and thereby secure not only low costs but uniformly satisfactory product.

Dr. N. S. Keith, Philadelphia, Pa.:—Is the material magnetic, or attracted by the ordinary permanent magnet? I believe the material would be affected by strong magnetism. I would like to know whether it can be attracted or deflected under ordinary permanent magnetism.

J. L. W. Birkinbine, Mexico City, Mexico:—I made a trip to Birdsboro, and I was interested in the product, and tried the effect of an ordinary horse-shoe magnet on it, both on the product from the flue-dust, as sintered flue-dust, and also on some magnetic concentrates. I spoke to Mr. Dwight this morning, and said that the sintered magnetic concentrates did not seem to be as magnetic as the unsintered material, and I desired an explanation, which he could not give me at the time, but still the product was sufficiently magnetic to be attracted by an ordinary magnet.

There was one point which Mr. Vogel brought out, that I desire to speak on, and that is the point in connection with the sintering, whether it was due to some form of fluxing or fusing, and I am inclined to believe his specification is correct, as the report of the attendant in charge—Mr. Dwight was absent at the time—was to the effect that it required more fuel to sinter a magnetite concentrate than was required for flue-dust, and I promptly assumed that the cause of this was that the flue-dust carried considerable sulphur, and was bound together by the partial fusing of silicates, while the magnetic concentrates

really carried very little gangue, and required some melting of the ferric oxide.

PROF. H. O. HOFMAN, Boston, Mass. (communication to the Secretary *):—In the discussion of sintering fine iron-ores the question of what caused the agglomeration of particles was considered in a general way. The following notes contain some thermal facts bearing upon the subject.

The leading factors in the sintering of ores are temperature and composition of charge.

As regards temperature, A. S. Dwight stated to me that in sintering finely divided iron-ore or flue-dust, with the Dwight-Lloyd machine, the heat was apparently about the same as in blast-roasting sulphide ore. Laboratory-experiments carried on with the Savelsberg process[1] in 1907 have shown that the highest temperatures obtained range roughly between 1,000° and 1,200° C.

The ore-charge contains iron oxides, gangue, and coke. Ferric oxide fuses, according to Kohlmeyer,[2] at 1,565° C.; that is, at temperatures much higher than that reached in the sintering process. There must be present a binder for the iron oxides, and this can be either a ferrite or an iron silicate. The only ferrites that need consideration are calcium ferrites. Kohlmeyer and Hilpert,[3] who traced the freezing-point curve, found that while with some mixtures of CaO and Fe_2O_3 sintering was noticeable at 900° C., the practical temperature lay at about 1,200° C. This temperature furnished also the lowest liquid fusion, namely, that of a mixture of 1 mol. CaO and 1 mol. Fe_2O_3, 26 per cent. CaO and 74 per cent. Fe_2O_3. Calcium ferrites may therefore act as binders at sintering temperatures.

As regards silicates, I determined,[4] in 1899, the formation-temperatures of various ferro-calcic silicates. The pure ferrous singulosilicate forms at 1,270° C., the sesquisilicate, $3FeO.2SiO_2$, at 1,140° C., the bisilicate at 1,110° C.; by substituting CaO for FeO, the formation-temperatures are lowered to the minima,

* Received Mar. 7, 1912.

[1] Hofman, Reynolds, and Wells, *Trans.*, xxxviii., 126 (1907).

[2] *Metallurgie*, vol. vi., No. 10, p. 323 (May 22, 1909).

[3] *Metallurgie*, vol. vii., No. 7, p. 193, and No. 8, p. 225 (Apr. 8, 22, 1910).

[4] *Trans.*, xxix., 682 (1899).

1,130°, 1,070°, and 1,030° C., respectively, when by further substitutions the temperatures rise again. The minima lie well within the ordinary ranges of temperature obtaining in the sintering of finely-divided ore.

The conclusions to be drawn are, that in the sintering processes the agglomeration of the ore-particles is due mainly to the formation of ferrous silicate, and to a smaller extent to that of a calcium ferrite.

Sintering and Briquetting of Flue-Dust.

Discussion of the paper of Felix A. Vogel, p. 381.

DR. F. W. C. SCHNIEWIND, New York, N. Y.:—Mr. Vogel speaks of briquetting the flue-dust by means of lime. I learned recently in Europe of a process employed with considerable success at one of the blast-furnaces, where blast-furnace cinder is used with lime. The lime and cinder are mixed with the flue-dust, and exposed to superheated steam, which converts the blast-furnace cinder into cement. Briquettes are then formed in the usual way.

MR. VOGEL:—The process referred to is the "Scoria process," which has not been in use for some time past. For that reason, I did not mention it.

PROF. ARTHUR L. WALKER, New York, N. Y.:—I would ask the commercial name for magnesium chloride, and where it can be obtained?

MR. VOGEL:—I mentioned magnesium chloride, because it has been used abroad to a large extent. In Germany it is obtained as a by-product in the manufacture of potash-salts; in the United States we have been using chloride of lime, which is cheaper.

OTTO SUSSMAN, New York, N. Y.:—What is the cost of this process?

MR. VOGEL:—The cost is exceedingly low. I can give you the figures from abroad, but not for the United States, because the system has not yet been used here commercially to

any considerable extent. In the experimental plant, the cost is somewhat high. In Germany or Belgium the cost of the process is about 1.60 marks per metric ton; at some plants it is 2 marks, so we may estimate it as from 40 to 50 cents. In the United States, our present experience is, that it will cost from 20 to 30 cents per ton. The operating-expenses are very low, because of the simplicity of the process.

ARTHUR S. DWIGHT, New York, N. Y.:—What basis is there for assuming that the chloride of lime or the magnesium chloride has a catalytic action, and, if so, what is the action?

MR. VOGEL:—We consider magnesium chloride to be a catalytic agent, because it persists after causing the chemical reaction to take place, which follows its addition to the oxides; in fact, it can be extracted from the hardened briquette by means of hot water, and is often found as a white efflorescence covering the surface of the briquette.

MR. DWIGHT:—The paper says that the briquettes become hot immediately after pressing. Is there any explanation why that occurs?

MR. VOGEL:—There is a very strong chemical action—in fact, there are several different actions taking place. One, a cementation between silica, lime, and alumina; another, the reoxidation of the lower iron oxides, and still another, the hydration of oxides. If hydration alone took place, the briquettes would disintegrate in the furnace, but Schumacher briquettes will not disintegrate until they reach the fusion-zone of the blast-furnace.

MR. DWIGHT:—Does not the heating action take place almost instantaneously, the moment the briquette is compressed? I understand from the paper that such is the case.

MR. VOGEL:—Yes, when you take hot flue-dust, but not when you take cold flue-dust. In the latter case the reaction starts a quarter or a half hour after the briquette has been pressed.

MR. DWIGHT:—I drew my conclusion from Professor Richards's paper also. Possibly it would be well to leave this point for future discussion.

B. G. KLUGH, Birdsboro, Pa. (communication to the Secretary *):—In Mr. Vogel's description of the Dwight & Lloyd process, he makes the following statement: "The resultant sintered material is not homogeneous, and, while a large portion of it is of cellular structure, it is glazed on the surface, which makes it quite difficult to reduce in the blast-furnace." This statement is so completely at variance with the facts that it is necessary to restate the properties possessed by Dwight & Lloyd sinter.

All sintering, agglomerating, and briquetting processes treat finely-divided materials which are readily mixed to any required degree of homogeneity by various well-known mixing-mechanisms. Any lack of homogeneity in the finished product will be due to some inherent feature in the agglomerating process which tends to destroy the uniform distribution of particles in the raw mixture. In the Heberlein process the heavy pressure of the upward air-blast keeps the particles of the charge at the upper surface in motion and also disturbs the particles surrounding every blow-hole and crater formed in the charge. As a result of this action there is a re-arrangement of particles in portions of the charge, and a corresponding lack of homogeneity in the finished product. In the Schumacher process or in any briquetting process the extent to which the particles of the homogeneous raw mix are re-arranged depends upon the pressure used in forming the briquettes. With great briquetting pressures, and with wide differences in shape and size of constituent particles in the material, there is a considerable re-arrangement of particles during the briquetting operation, and as a result some lack of homogeneity in the finished product. In the Dwight & Lloyd process the thoroughly-mixed charge is placed on the grates and sintered under conditions that maintain absolute quiescence of the particles with relation to each other, and absolute homogeneity is therefore one of the inherent properties of the finished product.

The Dwight & Lloyd sinter is a product of incipient fusion. If we take mill-cinder as an example of a product of complete fusion, and rotary-kiln nodules and Heberlein agglomerates as examples of products resulting from an intermediate degree of fusion, then the Dwight & Lloyd sinter represents the product

* Received May 7, 1912.

obtained from the least degree of fusion that effectively binds the particles together. During the sintering operation, wherever a particle of carbon reacts with a particle of iron oxide, there is formed a microscopic gas-bubble which in escaping through the semi-fused but plastic sinter forms a capillary pore. The progressive union of these minute gas-bubbles into larger and larger units produces a series of cells, canals, and pores all intercommunicating, and of all sizes from that of a pin-point under 40 diameters magnification up to 0.5 in. or more in actual diameter.

The typical structure of the Dwight & Lloyd sinter is similar to that of a very porous sponge. The cell-walls of the larger pores in the sinter are very thin, but are nevertheless highly porous, due to the presence of many microscopic pores, the number of which is roughly equal to the number of reacting particles of carbon in the sintering-mixture. The great reducibility of this sinter, which has been demonstrated wherever it has been used, is undoubtedly due to its remarkable intercommunicating system of pores and cells, as a result of which all of the cell-walls are in effect highly pervious diaphragms. It is thus evident that even if Mr. Vogel's statement that the surfaces of the sinter are glazed, be true, this condition would not affect the reducibility of the sinter since the glazed surface would be perforated at innumerable points by microscopic pores. It hardly seems possible, however, that Mr. Vogel could have used "glazed" in the sense in which it is ordinarily used. In the ceramic and petrographic arts a "glaze" is understood as a covering of glass. Glass is a non-crystalline product of fusion. In the Dwight & Lloyd sinter the formation of glass would require a degree of fusion and consequent fluidity that would entirely obliterate the innumerable microscopic pores which are typical of the structure.

The micro-photographic sections, Figs. 1, 2, and 3, show the Dwight & Lloyd sinter magnified 40 diameters. The phenomenal porosity is plainly shown, together with the great preponderance of dark opaque iron oxide, and the practical absence of the lighter colored silicates, either in the form of quickly-cooled glass or slowly-cooled crystals. The mere presence of so many capillary pores under high magnification proves conclusively that fusion did not extend beyond the stage of incipi-

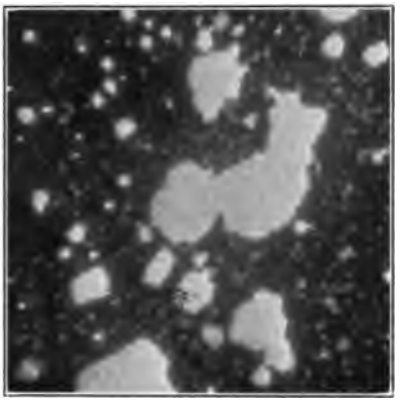

FIG. 1.—DWIGHT & LLOYD SINTER FROM FLUE-DUST AND MAGNETITE. MAGNIFIED 40 DIAMETERS.

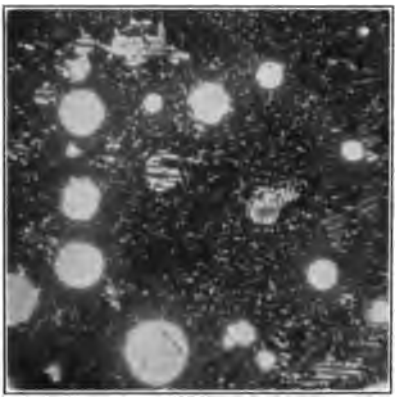

FIG. 2.—DWIGHT & LLOYD SINTER FROM FLUE-DUST AND PYRITES CINDER. MAGNIFIED 40 DIAMETERS.

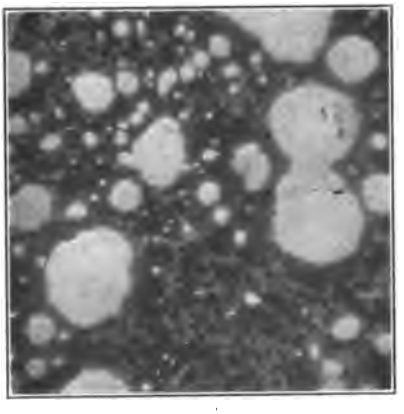

FIG. 3.—DWIGHT & LLOYD SINTER FROM MAYVILLE ORE. MAGNIFIED 40 DIAMETERS.

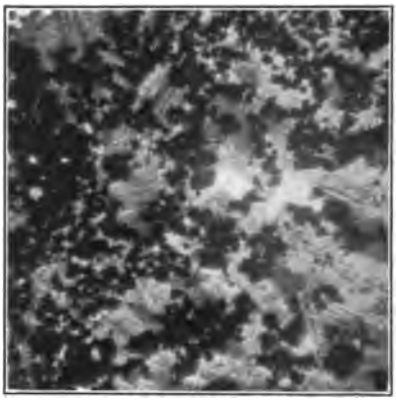

FIG. 4.—HEBERLEIN AGGLOMERATE FROM PYRITES CINDER. MAGNIFIED 40 DIAMETERS.

FIG. 5.—SAGMANEIRA NODULES. MAGNIFIED 40 DIAMETERS.

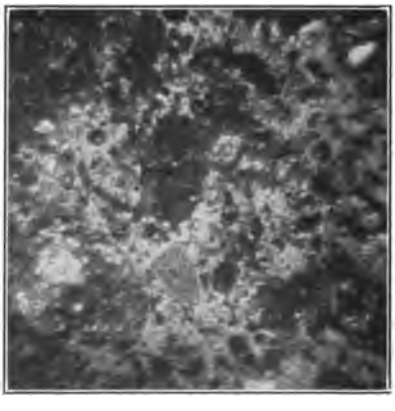

FIG. 6.—AMERICAN SINTERING CO.'S FLUE-DUST NODULES. MAGNIFIED 40 DIAMETERS.

ency. Fig. 4, a Heberlein agglomerate made from pyrites cinder, and Figs. 5 and 6, rotary-kiln nodules (all of 40 diameters magnification), show the absence of the typical cell-structure found in the Dwight & Lloyd sinter, and a great preponderance of fused silicates both in crystalline and in glass form. All three of these figures show a marked resemblance to the puddle-cinder of Fig. 3 in my paper presented at the New York meeting, February, 1912.[1] The presence of such a large percentage of light colored silicates, with a well-developed crystalline structure, conclusively proves that Heberlein agglomerates and rotary-kiln nodules, in common with puddle-cinder, have passed through a stage of almost complete fusion. That material possessing these characteristics is not readily reducible is demonstrated by the fact that the use of more than 15 per cent. of Heberlein agglomerates in a blast-furnace charge gives a scouring slag.

Briquettes have the type of porosity which is exhibited by water absorption and retention, caused exclusively by capillary action. Briquettes lack the intercommunicating system of cells and pores found in the Dwight & Lloyd sinter, and which are needed for the access of gas to the inner structure. The ineffectiveness of the pores in briquettes can be better appreciated by considering what an enormous pressure would be needed on one side of a briquette to produce an actual flow of gas through the structure. The porosity of a briquette is therefore of slight value in the blast-furnace, and in the stock-pile such porosity is an undesirable property, because water absorption, followed by freezing, entirely destroys the structure.

Wholly aside from any evidence here adduced, the Dwight & Lloyd sinter has been proved by actual use to be both homogeneous and very reducible. It has likewise been demonstrated that when 20 per cent. or more is used in a blast-furnace charge no scouring tendencies develop in the slag.

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[1] This volume, p. 367.

The Schumacher Briquetting Process.

Discussion of the paper of Joseph W. Richards, p. 387.

PROF. J. W. RICHARDS, South Bethlehem, Pa. (in reply to several questions):—The cut representing some of the round briquettes which were under test shows that, at the breaking-stresses, they break into lumps, and do not disintegrate into powder.

The process is simplicity itself. I have followed the different processes for sintering iron-ores and flue-dust, am interested in all of them and wish them all success; but this process appealed to me strongly by its simplicity. It is in line with a method which I once tried to work out myself, but which never came to anything, namely, the agglomeration of fine iron-ores by means of Portland cement. As small an amount as 5 or 6 per cent. of Portland cement will enable fine ore to agglomerate and set into a briquette, but when I came to estimate the cost, I found it was a little too high.

The Schumacher process, curiously enough, does not work well on iron-ore, but flue-dust thus treated does harden and give a strong briquette. In my attempts to find a reason for this difference, it has occurred to me that possibly the particles of flue-dust had been coated over with some sublimed silica, which might be an agent in the setting. But I found by experiment that neither sublimed silica, nor ordinary silica, nor ordinary alumina, would briquette. Chemically pure iron oxide briquetted quite hard; but the application of pressure is a necessary part of the operation. If simply moistened with the solution and allowed to stand it does not set hard, but the more pressure applied to it the harder it sets. Evidently the flue-dust particles must be pressed very close together in order to get the small proportion of active material wet, so that the particles shall act at their points of contact; for it is evident that only at the points of contact does the cementing action take place.

The reactions are facilitated by considerable pressure, bringing the acting particles close together, and may go on with

considerable rapidity, and thus a small amount of calcium chloride produces a considerable amount of bonding material at the points of contact of the particles, the bonding material being principally ferric hydrate.

Some think that an iron hydrate will naturally go to pieces when heated, but it does not if you start with lumps; and we know that these briquettes do not go to pieces when heated. It is true that we have other things besides pure iron oxide in flue-dust. There may be deposited or sublimed silica on the surface; and there are other materials mixed in with it, and these may combine with the iron hydrate to form a little different compound from simply pure ferric hydrate.

As to the success of this process abroad, I understand that there are 18 plants now in operation in Europe, and there has been no question as to the briquettes behaving properly in the blast-furnace. The briquettes are this week being worked in the blast-furnace at the Cambria Iron Works, and have proved their suitability.

A. S. DWIGHT, New York, N. Y.:—The increase in temperature shown by the fresh briquettes seems to indicate some interesting reaction, which, if understood, might throw light on the chemistry of the process. In the reactions which Professor Richards has written on the blackboard I should expect, without having actually figured it, that the thermal result would equal zero.

PROFESSOR RICHARDS: The sum total of the reactions is that the iron oxide becomes hydrated oxide, and the heat of hydration of the iron becomes the source of the heating of the briquettes. As the briquettes come from the press at Johnstown, they steam very vigorously. Part of the steaming was due to the briquettes being warm when made, and coming in contact with the cool atmosphere outside; but it is a fact that briquettes made of cold flue-dust, with calcium chloride, warm up afterwards on standing in the air.

The press does mechanical work upon the briquettes, which I calculated roughly could raise the temperature of the briquettes 10° C., but there is undoubtedly a heating effect due to chemical action, which shows itself after the making of the briquettes, while they are standing in the cars.

A. S. DWIGHT:—Returning to the matter of the increase in temperature of the fresh briquettes, I call attention to a phenomenon I have frequently observed in lead-smelting practice, which may perhaps throw some light on this question.

About 20 years ago briquetting began to be adopted on a large scale by the lead-smelters for handling fines of all kinds, such as roasted ores, flue-dust, and other by-products. This custom has now been largely abandoned on account of the high cost and generally unsatisfactory metallurgical results obtained with the briquettes. The practice was to mix the material thoroughly with from 5 to 10 per cent. of milk of lime, as a binder, and press it into briquettes, using any of the several forms of apparatus developed for the purpose, the most successful of which was the "White" mineral-press. The briquettes as turned out were delivered automatically to a traveling-belt, from which they were picked by hand, stacked, and air-dried until they would stand handling to the furnaces. When flue-dust was present in the mixture, there was always observable in the neighborhood of the mixer and press an odor of ammonia, sometimes so strong as to irritate the eyes of the attendants. At the same time the briquettes exhibited a perceptible rise of temperature, similar to that described by Professor Richards as taking place in the Schumacher process. Just what the reaction may have been which caused the evolution of the ammonia, I would not care to say definitely, as I never seriously investigated the question. The flue-dust from a lead blast-furnace is likely to carry from 1 to 5, probably averaging 3, per cent. of carbon as fine coke-dust, besides iron oxide, silica, limestone, lead oxide, sulphur, etc. With coke-dust, iron oxides, moisture, and active lime-reagents present in both cases, there certainly seems to be a sufficient similarity to lead us to suspect that one reaction might throw some light on the other. I simply suggest this thought as a possible help in the inquiry to explain what actually takes place in the Schumacher process—a problem which at the present time seems to me not satisfactorily solved.

PROF. HENRY M. HOWE, Bedford Hills, N. Y.:—The explanation which Professor Richards has given seems to be reasonable, but I am not sure that it would not be more convincing if

we left out some of the reactions. If we start with calcium chloride and iron-ore, it is reasonable to suppose that we may end with a large amount of basic iron chloride. We know how basic iron chloride (as, for instance, in a rust-joint) makes a cement—that is what the stuff really is. We can get such a cement with a little sal ammoniac and iron-filings—according to my recollection, a sticky sort of cement containing basic iron chloride. From a small amount of chlorine we get a relatively large quantity of this sticky basic iron chloride. The reason I think the reaction is likely to occur in that way, rather than by the exact steps mentioned by Professor Richards, is that the first reaction looks as if it would be an endothermic reaction.

As between iron-ore and flue-dust, is not the reason why the former will not agglomerate, or cement, as the latter will, that iron-ore is very much coarser than flue-dust? As I understand it, that which cements in a roadway, for instance, is the very finely powdered trap-rock, and if you have rock which is not broken up into a finely-powdered form, you will not get any cementing action in the road. That may be one of the conditions in this case.

As to the heat given off, the amount is not stated. If it is considerable, it will have to be made up for, in the furnace. For it implies a loss of energy, which must be replaced in the furnace by a correspondingly greater evolution of heat. I fancy, however, that this item is not important.

As to the mechanical differences in these briquetting processes, it is very easy to say that the mechanical conditions are all right or all wrong, but so long as these assertions are merely speculative, they ought not to carry great weight. We must look to actual practice. As I understand the author of the paper, he has investigated this matter and is convinced that there is a certain reduction in fuel-consumption through a given process. A piece of actual evidence to that effect is worth all the speculation that can be had as to whether the result ought to be one way or the other.

Dr. F. W. C. Schniewind, New York, N. Y.:—Perhaps Professor Richards could throw some light on this matter by stating what he knows about the thermo-chemistry of the re-

action that begins with ferric oxide and ends with ferric hydroxide. How many heat-units are involved in the conversion of the oxide to the hydroxide? Furthermore, has Professor Richards made any investigation as to whether any of the other ingredients of the flue-dust alone, with calcium chloride or magnesium chloride, will show heating under compression?

PROFESSOR RICHARDS: I have tried the other ingredients of the flue-dust, such as silica and alumina, in the laboratory. When moistened with the solution they became cold, and under pressure (on a small scale) they did not show any perceptible rise in temperature. With regard to the operation on a larger scale, the ore of Johnstown was quite cold, the flue-dust was warm, but not hot, and the briquettes coming out of the press seemed to the touch much hotter than they should be from the admixture of the two materials—30 parts of flue-dust and 70 of ore. I believe that when the operation is carried out on a large scale there is always a warming-up of the briquettes, which is greater the warmer the mixture before being pressed.

As to the heat of hydration of iron oxide, I have figured that it would raise the temperature of the briquettes about 80° C., if all the iron oxide present were hydrated at once.

A. H. COWLES, Sewaren, N. J.:—I can appreciate Dr. Richards's evolution of his hydrolysis theory, from the fact that if you start with aluminum chloride solution, it does hydrolyze, and magnesium chloride also tends to separate out chlorine quite easily, but calcium chloride less than either one of them. Dr. Richards, being so familiar with the chemistry of aluminum, I can see easily the way that theory would arise in his mind, as an explanation of the so-called "catalytic effect" of the salt.

PROFESSOR RICHARDS: In answer to Mr. Dwight's question as to why he should have obtained ammonia by wetting fume from lead-furnaces, it is possibly from nitrides of some of the materials which are blown out of the furnace, particularly aluminium nitride. There are processes for making aluminium nitride by heating alumina with carbon in a furnace at the proper

temperature and blowing nitrogen over it, and that nitride, when moistened afterwards, would give rise to ammonia.

A. S. DWIGHT: Would that be likely to occur in the case of lead-furnaces, in which the temperatures necessarily have to be kept much lower than they would be kept in an iron blast-furnace, and the flue-dust of which undoubtedly comes from near the upper zones of the furnace, where the temperature is so low that you can hold your hand there without discomfort? The flue-dust comes principally from the upper 3 or 4 ft. of the lead blast-furnace.

PROFESSOR RICHARDS: The lead-fume does not all start from the top of the furnace; part of it starts from the lower portion; and I suggest the possibility of nitrides coming from that portion. It takes only about 1,200° C. to begin the production of aluminium nitride from a mixture of alumina and carbon, through which is passed a gas containing nitrogen and carbon monoxide.

The Briquetting of Iron-Ores.

Discussion of the paper of N. V. Hansell, p. 394.

In connection with the presentation of his paper in oral abstract, Mr. Hansell exhibited samples of briquettes, showing their porosity, and the change of surface produced by burning.

ALFRED H. COWLES, Sewaren, N. J.—I am familiar with this furnace, and I wish to point out that the larger it is, the greater is its efficiency, and the longer its life. In Europe, experiments have been made with small, short furnaces of this character, accommodating narrow cars; whereas Mr. Hansell says that there is in this country a furnace 180 ft. long and 10 ft. wide. The American furnaces, moreover, differ in design from those used in Europe; and we may expect from these larger furnaces not only greater efficiency, but also greater durability —the latter advantage being due to the fact that, in these large furnaces, projecting brick walls of greater thickness can be provided, to prevent the heat from attacking the sides of the cars, by means of bricks which, over-reaching these iron surfaces, furnish a protective cover to them. In other words, in these

large furnaces, the iron sides of the cars, slipping through under the side-walls of the furnace, are more effectively protected than in small furnaces, against the action of the heat.

I think we may expect that, in large furnaces, such as are building in this country, the great mass of magnetites can be effectively treated at low cost, and with small consumption (probably not more than 4 per cent.) of fuel.

[Mr. Cowles mentioned also the diminution of phosphorus and sulphur in the products of smelting the briquettes, as shown by the analyses presented in the paper; but Mr. Hansell declared that the briquetting process did not eliminate phosphorus, and explained that the difference in phosphorus shown by the analyses between the crude ore and the briquettes was due to an intervening concentration of the ore, before it was briquetted. This explanation, which was at once accepted by Mr. Cowles, is published simply to prevent any similar natural misinterpretation of the chemical analyses referred to.]

The Refining of Blister-Copper.

Discussion of the paper of Horace H. Emrich, p. 446.

ALBERT R. LEDOUX, New York, N. Y.—I can add a little to the information which has been given, by saying that it has been our business for some years, among other things, to inspect outgoing copper from some of the works here in the East, and to find out why there are variations not only in the conductivity of the wire-bars, but also in the character of the cathode-copper shipped; and we have found two things: These impurities which have been mentioned are largely upon the surface of the cathode. Samples taken of the inside, even of the nodules or "berries" which grow on the outside of the cathode, show that they do not contain more impurities than the denser material in the interior. In other words, it is a superficial deposit or contamination. There is still another reason for impurity in some of the copper which finds its way to the manufacturers, namely, carelessness in the handling of molds or in casting. We have found copper which was comparatively pure, as far as the copper itself was concerned, but contained, mechanically intruded in the interior of the ingots,

cakes, or wire-bars, impurities which would lower the quality of the whole.

As a fact bearing on that, I may say that we have noted in our laboratory, within the last few days, some interesting points in connection with experiments which we have been making. A very large manufacturer had had some shipments returned to him because the metal, an alloy, would go to pieces under stress or friction; would flake off or break along certain lines when in use. Analyses of samples of the metal, taken by boring through the whole mass, did not show any appreciable impurity. Nevertheless, there was this flaking off; and we were asked to ascertain the cause. By planing down some of the metal, we found layers or streaks of dark material, which was proved, both by analysis and by microscopic tests, to consist of graphite. We learned that the molds had been made in damp sand and then lined with graphite. But the graphite was found later in the interior of the casting, not on the outside —a fact which puzzled us for a long time. We think we have hit upon the explanation: that the heat of the molten metal turned into the mold generated steam in the wet sand; that this steam had no means of escape except towards the surface, and found its way up through the metal, carrying along, mechanically, a part of the lining of the mold.

I am tempted to tell of two other experiences, showing the difficulties under which the most scientific managers of electrolytic plants sometimes labor.

Some years ago, the head of one of the most important electrolytic refineries in the United States was puzzled to discover the cause of certain short-circuits. He noticed that an accretion started at the anode and grew always from one side towards the cathode until it had completed a short-circuit. The works ran north and south, and this short-circuiting was always from the west towards the east. The manager came very near making a statement before a scientific society that the revolution of the earth must have something to do with this short-circuiting, as it always was in the direction from the west towards the east; but one day he broke off one of these fingers, which ran across, and found, at the point where it began, a little splinter of wood. He broke off others and found the same thing; and, to make a long story short, the

reason was just this: The anodes, which were heavy, were brought into the factory, unloaded from a car, and dumped by the men on a block of wood alongside of the tank. They did it always the same way; a corner of the anode always struck the wood at a certain inclination, and little splinters of the wood were broken off, and adhered to the bottom of the anode. When the anode was put into the tank (always in the one way), that splinter served as a starting-point for a short-circuit, and that was the simple solution of something that had been bothering them for more than a year. When they substituted a rubber mat for a wooden block, the trouble ceased.

But not all the difficulties of owners of electrolytic refineries are due to such accidents. We were retained some years ago by the owners of a large plant to ascertain what caused their extraordinary silver-loss. They analyzed very carefully their ores and the materials which went into the electrolytic refinery. They kept very close watch of all the operations, and yet there were losses in silver which they could not locate. The technical man in charge of the electrolytic refinery claimed to possess certain secrets, employed in the process; and it was agreed in his contract that no one should be admitted into the room where the purifying of the electrolytic solutions took place under his sole charge. By the way, we were told that he also put something into his tanks, which he bought himself, and kept as a secret. We succeeded in getting hold of some of the material and found it was nothing but anilin green. It gave a brilliant color to the solution; that was all.

We became convinced in our investigation that there was something wrong with the purifying-room; that the silver that went into it did not come back into the tanks. We took some men one night and dug a trench around the end of the house, on the side towards the town sewer. We discovered the original solution running down this sewer! The secret processes of purifying consisted in part, at least, in wasting this valuable solution, containing very large quantities of silver. Of course he could make his solution pure by renewals, after letting the foul solutions run to waste in the sewer carrying silver with them; and that is where the loss in silver came in in that particular refinery.

DR. RAYMOND (in reply to a question): Mr. Emrich does not say how he proceeded in separating and drawing into wire a sample of the cathode to which he refers as giving, under that test, a high conductivity. He speaks of the impurities as mechanically deposited in the cathode, and sometimes as being laminated. He is talking about a laboratory specimen, and I suppose he took a small piece which was apparently free from impurities.

DR. N. S. KEITH, Philadelphia, Pa.:—We know that electrolytic deposits are not of a strictly reguline character; they are simply agglomerations of minute crystals. How coherent they are has not been determined. Attempts made to work such copper, without heat enough to cause fusion and form alloys of copper with the materials inclosed between the crystals, have been unsuccessful for producing metal to be drawn into wire.

The statement of Mr. Emrich, that frequently, from various causes, wires drawn from the wire-bars made by the fusion of such cathodes possess less conductivity than is commercially desirable, is undoubted; and it is most probable that impurities, like arsenic, sulphur, antimony, etc., carried over mechanically by agitation of the electrolyte, have contaminated the original cathodes. Various efforts have been made to prevent this transfer, such as inclosing the anodes in bags, to act as screens; but hardly any bag we can make for practical use will prevent the passage of exceedingly minute particles which will float in the electrolyte, in which they may be present, even when the solution looks perfectly clear. I learned that from my own investigations some years ago, and I also found that if the electrolyte in direct contact with the cathode contains copper enough, then, under proper intensity of current, copper will be deposited without the other dissolved metals. If the voltage rises somewhat, and if there is a deficiency in the amount of copper ions in direct contact with the cathode, then hydrogen is deposited with copper, because it is the most prevalent ion in the solution, and will be occluded in the exceedingly minute interstices between the crystals of the deposit. Agitation of the electrolyte is necessary to replace continually the solution which, in immediate contiguity with the cathode, has become depleted of copper, with solution containing the normal amount of copper

ions. Hence, the minute floating particles of slime might also be included in the cathode through mechanical deposition.

The important question is, What improvement in practice may come from these observations? Mr. Emrich's production of good wire from a specimen of a bad cathode is, of course, not a proof that such cathodes can be separated in practice, without melting, into good copper and impurities. What he argues is that the impurities in such cathodes are mechanically mixed and not alloyed; hence, that they are not due to the electric deposition; and, therefore, that their source is the anode, not the electrolyte, and that remedies must be sought in the direction thus indicated. His observations should certainly be tested in the research-laboratories of our great electrolytic refineries.

ANTON EILERS, New York, N. Y. (communication to the Secretary*):—I do not think there is anything to be added to this paper at the present time. The facts as given are no doubt correct, for I know from personal contact that Mr. Emrich was a close observer and a logical thinker.

But this paper carries a suggestion to those who work in this particular line. The subject is by no means exhausted. It should be the ambition of some investigator to find out definitely, why, when slimes are caught as such in the copper during its deposition, the conductivity of the copper is not impaired, if the tests are made directly from the cathode, while it is lowered to a marked degree after the same cathode-copper has been melted down, even if an attempt to refine it further by oxidizing-heat has been made.

To me this behavior of the cathode-copper before and after melting indicates that the impurities in the slimes are present in a different state before and after melting down, and that the arsenic and possibly the antimony present play an important part. I would therefore suggest that a number of complete analyses of the slimes afloat in the first overflow from the tanks should be carefully made, particularly to find out in what combination, if any, arsenic and antimony are present; that another number of complete analyses of the melted cathode-copper be made for the same purpose, and that these analyses be supple-

* Received Mar. 22, 1912.

mented by metallographic tests of respectively the unmelted and the melted cathode-copper. The results would probably indicate steps to be taken to reduce the difference in conductivity to a considerable degree. Without the above, and other tests resulting from these, we are at present in the dark. Yet it is clear that large commercial advantages may be gained if the above questions are favorably settled.

Electrical Fume-Precipitation.

Discussion of the paper of F. G. Cottrell, p. 512.

F. W. C. SCHNIEWIND, New York, N. Y.: Have you any idea as to cost of operation?

MR. COTTRELL: I omitted to speak of the cost of installation. The precipitation-plant at the Balaklala smelter, up to the time of initial operation, including the construction of flues, installation of the apparatus, etc., cost $107,000, and subsequent modifications probably represented at least $15,000 more. Upon the information gained in this plant, the installation at the cement-mill, handling nearly twice as much gas, has been reduced in cost in a number of ways, but the overhead construction, carrying the precipitator at the top of the stacks, is naturally expensive, yet, even so, the total cost of construction will probably not greatly exceed $150,000.

The mechanical details of the tubular form last shown have been worked out more recently than those of the plate form, and the tubular form promises to be still less expensive in construction, as you would naturally expect. The cost of operating the plant varies somewhat with the character of the gases used. The amount of power consumed is not determined by the tonnage of the material collected, but by the volume of gas to be handled. It takes as much power to precipitate a thin cloud as a dense one.

In the cement-plant they are getting a higher efficiency; *i. e.*, a greater number of cubic feet treated per kilowatt than in the earlier plants. This is perhaps partly due to higher temperature of the gases, which here average about 450° C. The best figures on the cement-plant are between 3 kw. and 4 kw. of

electrical energy for the unit shown in the picture, which is an entirely insignificant quantity of energy for the amount of gas treated, representing for the complete installation probably less than 1 per cent. of the power used for other purposes about the mill. The new units being larger, however, will consume somewhat more energy.

DR. SCHNIEWIND: What is the tonnage production?

MR. COTTRELL: The mill is rated at 4,000 barrels of cement per day. Smaller installations would naturally consume somewhat more energy per cubic foot of gas treated. In the Balaklala plant, the power-consumption was about 120 kw. This higher power-consumption was partly due to the less perfect state of the art and partly to different conditions of the gases under treatment.

E. GYBBON SPILSBURY, New York, N. Y.: I wish to ask about the coating of the plates with deposit; first of all, how this is thrown down, and also whether, after long use, there is not a gradual building up, hardening, and attaching to the plates themselves, of a coating which might eventually choke up the whole thing, and cause those products, which are deposited at so great a heat that they would almost fuse together, to be an obstruction in the operation of the plant. In the flues there would be an aggregation of flue-dust which would gradually cement together.

MR. COTTRELL: That is one thing that has to be met very carefully in the engineering of the plant by determining the proper temperatures at which to install the apparatus. The plants thus far installed have furnished an agreeable surprise by presenting less of these troubles than was expected. The first plant treated simply sulphuric acid fumes, which became liquid and ran off the plates. The next was in connection with roaster-gases at the Selby plant, where they were roasting a mixture of California concentrates. The fumes here were a mixture of sulphuric acid and flue-dust, with some arsenic and lead sulphate, and it was difficult to keep the apparatus from caking up. The work began with the idea of putting in iron plates in a brick flue. After making careful tests, it was de-

cided that, in order to accomplish what was wanted in the way of suppressing all the fumes, it would be necessary to cool the gases sufficiently to insure all the sulphuric acid being condensed from the gaseous state into liquid droplets. This made the precipitated mixture so muddy that it was necessary to abandon the iron and brick construction and substitute lead flues and lead electrodes in its place. In this case the material came down as a mud more or less caky. This necessitated the intermittent operation of individual precipitation-units. Each was operated about 6 or 8 hr. at a time, and then it was necessary to open the top of the flue on hinges like a box and flush the electrodes down with a hose.

It was found that as long as the materials thrown down did not carry more than 10 or 15 per cent. of moisture, the deposit was so spongy that it could be shaken off; but when, for example, the material contained equal proportions of sulphuric acid, moisture, and solids it became necessary to wash it off with the hose.

In the work at the Balaklala smelter variable conditions were encountered. It was at first thought there would be considerable corrosion of the iron flues and electrodes; but after six months' running no serious corrosion was apparent. The apparatus stood extremely rough handling.

In the cement-mills the conditions are again very different, there being no acid or moisture to fear. The conditions probably vary very much in different mills, but, in the Riverside mill at least, the dust comes down in very easy form to handle. They burn a raw mixture, which gives a slight excess of lime in the dust. When dusts with higher alkalies and higher alumina are dealt with, it may be necessary to study the conditions of temperature and composition of the dust more carefully to prevent accretions.

The converter-gas problem, at present, is purely a matter of saving values. There has been a good deal of difficulty in past years in finding methods to collect these particular fumes, which are often very rich in lead as well as in other values. The use of the electric precipitation-method in conjunction with the basic-lined converter gives good promise of opening an outlet for many siliceous, low-grade lead-ores which are at present

worthless, because they cannot be profitably treated in the lead blast-furnace.

MR. SPILSBURY: What is your method of shaking the plates?

MR. COTTRELL: Different forms of rattling- or hammering-arrangements have been employed to suit varying types of construction, a relatively slight jarring of the plates or tubes being usually sufficient, the dry deposits shaking loose very readily.

Fortunately, the work is now getting over on to the ground where it is being done from an economical standpoint rather than from the standpoint of possible litigation. I do not suppose, however, it would have been developed to the extent it has been, except from the pressure which was brought upon it to avoid litigation. It has gone beyond that point now, and is being developed in a much more methodical way on a commercial basis.

A. H. ELLIOTT, New York, N. Y.: In your main talk you said you were using oil for burning the cement. What would happen if the oil was in excess and you got smoky fumes—would the tarry matter prevent the action?

MR. COTTRELL: The soot is precipitated as well as the dust. They are putting an installation into one of the gas-plants in California to treat the smoky residuum from making illuminating-gas from crude oil. There is quite a field of application in that direction. Each one of these applications requires a certain amount of engineering adaptation, but the apparatus has proved itself flexible enough, with proper study of the details of application, to meet all the conditions.

DR. ELLIOTT: My idea was, that you would want to keep the apparatus clean. We have to consider also the hydrocarbon ingredients in the fumes from the gas of soft coal or oil fires. We have a number of oil-firing apparatus and the smoke is a great nuisance. The thought I had in mind is that it would be an insulator very sticky to remove.

MR. COTTRELL: I imagine in handling a tarry liquid mass you would have some difficulty, and you might have to get

over that by special means. The gas-works installation mentioned collects what is practically a clean lampblack or soft carbon.

I used to illustrate this process in a laboratory-demonstration by simply rigging up a tobacco-pipe and blowing the smoke into a glass precipitation-chamber, turning on the current and precipitating it in that form.

I always decry, however, the tendency to hold this system up as a general means of suppressing the coal-smoke nuisance. That should be done, in most cases, by proper combustion in the first instance. When you have good combustion you get rid of the smoke-nuisance. Still, there are isolated special cases where it may be more expedient to precipitate smoke electrically than to consume it.

L. C. GRATON, New York, N. Y.: I think, perhaps, the installation of Mr. Cottrell's which has received most attention from metallurgists is that at Balaklala. The fact that the courts closed that plant, after a trial of the process, has led some people to conclude that it was not satisfactory. From acquaintance with conditions in Shasta county, I think I can say something that possibly Mr. Cottrell did not feel free to say, and that is, that the so-called farmers of that section have for many years carried on a campaign of extortion against the smelters, and there is very little doubt in the minds of disinterested parties that, if the question had been settled on its merits, instead of by biased judges, the Balaklala plant would be operating to-day with Mr. Cottrell's system.

There is one question I would like to ask, not from a metallurgical standpoint so much as indicating the possible presence of certain constituents in the ore, and that is, whether such a thing as bismuth or arsenic was found in the fume?

MR. COTTRELL: That ore seems remarkably free from those heavy metal constituents. As you go up the "Copper Crescent," in Shasta county, there seems to be an increasing quantity of lead, arsenic, gold, and silver. I cannot cite, however, any detailed determinations. I have never seen any statement of the quantity of bismuth present. The lead at Balaklala was extremely low, but I cannot speak with any degree of certainty

of the average runs there. I was perfectly willing to keep out of anything that might draw me into the controvery as to what the deleterious constituents were. I was at Balaklala in the capacity of putting in a particular, definite technical device, and I wanted particularly to keep that separate and free from any question of fume-litigation, as far as I was concerned. For that reason, I did not take any special pains to inform myself on the constituents of the material, outside of those matters we had to deal with in our actual operations, but my impression was that the lead, arsenic, and bismuth were surprisingly low in the fume, compared with many other places.

DR. SCHNIEWIND: Has this method ever been employed for the removal of dust from iron blast-furnace gas for use with producer-gas in gas engines?

MR. COTTRELL: That application of it has just begun. Attention was at first centered on the most urgent questions in the West, where the apparatus was originated and developed. I did take up some of that work later, however, and discussed it with some of the representatives of the iron firms in Indianapolis and Chicago; and we had some tentative negotiations under way to take up work along that line, when the relations with the Smithsonian Institution developed, and made it inexpedient to get that work under way under the old management, and perhaps, later, have to make new arrangements; so it was thought better to defer the matter until the new plans were in definite shape. Somewhat similar work is, however, developing in the West in connection with producer-gas and oil-gas engines.

RICHARD PETERS, JR., Philadelphia, Pa. (communication to the Secretary *):—In the discussion of Mr. Cottrell's paper I note that Dr. Schniewind asks if the method described by Mr. Cottrell has ever been applied to cleaning gas in iron blast-furnaces. In answering, I beg to say that a patent was granted in England to Joseph Thwaite, but it has now expired. From the wording of his patent, it seems that he used it in connection with blast-furnace practice and obtained a partial separation of

* Received Oct. 3, 1912.

dust. His apparatus consisted simply of two vertical gas-flues for alternate operation; in the center of each flue was suspended a long barbed wire charged by means of a static machine. The current and gas were alternately passed through each flue and the dust precipitated on the sides. A quantity of dust adhered to the wires, which Mr. Thwaite claimed could be kept uncovered by breaking the circuit every 15 min. I am unable to say to what extent this apparatus succeeded, or why it was not adopted by others, but I should judge that the rapid flow of the gas past the barbed wire in the flue did not allow sufficient time to settle the dust in that manner, without the aid of centrifugal force and skimmers discharging into a separate dust-chamber.

A patent has recently been granted to Mark W. Johnson, Jr., of Birmingham, Ala., for an apparatus for the dry cleaning of gas.[1]

Mr. Cottrell states, on p. 518: "The precipitation of suspended matter in gases or liquids may be accelerated by either the alternating or the direct electric current. The former acts chiefly by agglomerating the suspended particles so that, constituting larger bodies, they settle by gravity more rapidly," etc. He has devoted his energies towards solving the problem by direct current which deposits the dust on plates or tubes, whereas Mr. Johnson has endeavored to apply the alternating current, bringing into play centrifugal force acting with gravity, it being possible to do this in blast-furnace practice, since the gases are blown out of the furnace under pressure at a very high velocity, by the blowing-engines. To utilize this high velocity for creating a maximum of centrifugal force, the gas is passed first through a nozzle, which increases its velocity without creating back-pressure, by the same principle as that used in the Venturi meter. From thence the gas is directed tangentially into a ring-shaped hollow chamber, into which it is forced to whirl very rapidly in the form of what is technically described as a "vortex ring motion." This motion is a natural form of movement for gas and is sometimes assumed naturally by it when ejected from locomotive-stacks or large exhaust-pipes from sta-

[1] For a further detailed description of the apparatus, refer to the patent reports, U. S. Patent No. 1,024,784, April 30, 1912.

tionary engines. The advantage of this form is that for a given volume, considering also length and breadth, it gives a minimum radius, and, therefore, higher centrifugal force, since centrifugal force increases as the radius decreases. Skimmers are provided which take off the outer stratum of gas and divert it into a dust-chamber separate from the vortex-chamber, into which the main current of gas cannot enter and which is, therefore, a "dead end," where the movement of gas is slow and the dust is precipitated. A return-pipe is provided which sends back to the vortex-ring chamber the gas skimmed off, after it has passed through the dust-chamber and has there deposited its dust. By this system of repeated skimming and returning, a concentrating process is set up, which reaches a maximum of dust-carrying capacity, above which the dust is obliged to settle out. A discharge-ring is placed in the center of flow of the vortex-ring, and is therefore at a maximum distance from the skimmers and dust-outlets, and also where the tendency of dust to enter the discharge-ring is met by a maximum resistance, owing to the decreasing radius as the center is approached, and therefore increasing centrifugal force. Alternating current from an ordinary generator with step-up transformer is introduced by a series of fine points borne in clusters on copper bars, and the body of the machine is grounded.

Alaska Coal-Land Problems.

Discussion of the paper of H. Foster Bain, p. 595.

R. W. Raymond, New York, N. Y.: Mr. Bain's introductory statement that there is a growing disposition on the part of the public "to change emphasis" in government from "political equality" to "social justice" is somewhat vague; but I gather from his argument that he approves the growing disposition to which he refers, and that the "emphasis" which it demands involves a radical change in the policy hitherto pursued by the United States—the policy under which the Pacific States and Territories have been developed thus far. I do not understand how "social justice" is to be furthered by the sacrifice of "political equality," or how the change is to be effected by "emphasis." But I fancy that the beneficent and necessary

new system of government which Mr. Bain has in mind is the administration of the natural resources of the public domain for the profit of the whole nation (*i. e.*, of the Federal treasury), instead of transferring them to individual citizens, or to the States and Territories in which they are situated, to be utilized as a source of growth and prosperity by those communities.

The general discussion of this proposed reform I have undertaken in a separate paper (announced for the San Francisco meeting, but withdrawn when Mr. Bain's paper was announced), which I hope to present at the next opportunity. In that paper, I shall argue that, in the light of the history of our public lands and our land-laws, a change of our policy at this time would be a handicap upon the progress of certain States and Territories, and would ultimately result in a Union of States, some of which, by reason of events in their history, could not exercise, as to their own natural resources, the powers secured to others. Another part of my historical argument consists in the narration of an experiment in the Federal administration of public mineral lands, undertaken under conditions much more favorable than would surround such a venture now, pursued for 40 years, and finally abandoned in despair, by the vote of an overwhelming majority of Congress, acting upon the recommendations of a Presidential message. In these days, when "emphasis" seems to have superseded even memory, that story may surprise many who think they are now proposing a new substitute for the old system which our fathers established, but which, after serving us well for a century or more, must now be pronounced inadequate—with due apology to the said fathers, who "did not foresee," etc. It is not fashionable to recall anything prior to the war of '61; some ardent reformers even seem to think that history, as a useful guide, begins with the close of the Spanish war, when the United States, having picked up the island of Porto Rico, the island of Guam, and the 1,500 islands of the Philippine archipelago, became a "world-power." To such I can recommend the story, which will at least show them that the plan which they regard as necessitated by new conditions, and suggested by a new generation of progressive statesmen, is simply an old failure, revived by ignorant enthusiasm. It is not "progress," but reversion to an obsolete ancestral type.

Another proposition defended in my paper is that, if governmental interference with private enterprise is warranted at all, it should be confined, so far as possible, to local governments. Apart from all questions of constitutionality, it is the verdict of a thousand years of experience that the government which regulates the affairs nearest to the citizen should be itself nearest to him. This I shall support in my forthcoming paper with instances from our own history and that of other nations; and I shall here offer, under this head, but a single remark.

Namely, mistakes in legislation, or abuses in administration, are much more easily exposed and remedied, if committed by the government of a single State or municipality, than if they occur as features of a national policy, and its enforcement by Federal Executive Departments. The wrongs suffered by the citizens of a given State, for instance, can be removed by arousing the sense of justice of the people of that State, and bringing to bear upon its Governor and Legislature their demand for immediate righteous action. But it is practically impossible to make the Congressional representatives in Congress of forty-eight States so hot with indignation over matters affecting the citizens of one State only as to unite them for immediate action.[1]

National evils of whatever kind can of course be ultimately cured; but local wrongs, such as I referred to, cannot be promptly corrected by Congressional action. This is simply the verdict of actual experience; and upon it is based the doctrine of "home-rule," which needs no further comment beyond an apology for re-stating and re-defending it before intelligent men. If it were not so old as to be immemorial and so true as to be trite, I might offer abundant proof of it in the wrongs recently suffered by American citizens, without practicability of immediate redress, from the arbitrary acts of Federal officials.

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[1] This is the statement which was reported by the correspondent of a technical journal as a declaration on my part that "you cannot cure national evils," and was characterized as "brutally pessimistic." It is difficult for me to understand how any intelligent hearer could have so misunderstood my declaration; and it is still more difficult to conceive how any man who knew me—whatever he might have thought he heard me say—could seriously ascribe to me a declaration so untrue and unnecessary, and so contradictory to the work of my whole life. In short, I am reasonably sure that I was incorrectly reported, and absolutely sure that I was grossly and unwarrantably misinterpreted.

But Mr. Bain need not have prefaced his paper with generalizations about government by emphasis, or criticisms of the inadequate foresight of the fathers, or declarations of the necessity of new methods to meet new conditions. For he proposes primarily nothing which is not wholly consistent with the notions of the said fathers, and with the Constitution which they framed. In other words, he simply suggests that the United States government, owning certain coal-lands in Alaska, and needing for its navy a safe, sufficient, and suitable supply of coal in its own territory, shall open a coal-mine in Alaska. The United States, as actual owner of certain coal-lands, can mine coal from them, if it sees fit to do so, just like any other private owner. Whether the enterprise would be wise from the standpoint either of business or of statemanship, may be a question. But, from the standpoint of constitutional law, it is perfectly legitimate, quite outside of all declamatory emphasis as to social "justice." It is true that Mr. Bain proposes, in connection with this coal-mining operation, further measures, such as the construction of a trunk-railroad, the leasing of other coal-lands to private corporations, and the organization of a selling syndicate, in which the United States government shall be a member. But these other measures might be undertaken, in substance, whether the government had a colliery of its own or not. The first question to be considered, therefore, is the advisability of the mining of coal in Alaska by the United States government for its own use; and this question is to be judged from the standpoint of an owner of Alaska coal-lands who happens to be also a large consumer of coal.

Where is this colliery to be? The best region for the purpose of naval supply is undoubtedly the Bering coal-field; but every piece of known coal-bearing land in that field is already located, and, as I understand, the rights of the locators have not yet been finally adjudicated. Some of them may be ultimately ousted; others may be confirmed in their titles. Is it certain that the lands of the ousted claimants will include the best location for the proposed government colliery? Should not the government, instead of starting a colliery on lands which it still happens to own, select the most favorable place, without reference to ownership, and buy the tract, if necessary?

The selection of a suitable location involves not only the

conditions of mining and transportation, but also the assumed quality of the product. This cannot be settled by the analysis of selected, or even of carefully averaged, samples taken at or near the outcrop. Nothing is more familiar to colliery-managers than the variation in quality—especially in percentage of ash—exhibited by one and the same coal-bed. And every consumer of coal who is also owner of a coal-mine knows that he may lose money by continuing to use his own coal, when it has temporarily deteriorated in quality, instead of selling it for whatever it will bring in the market, and purchasing better fuel for his own special purpose. I was for 20 years connected as engineer with a firm which owned coal-mines, iron-mines, furnaces, rolling-mills, etc., yet often found it economical to sell its own coal and ore, and to buy materials more completely adapted to the other departments of its business.

The plan of a government colliery to supply the navy with coal involves the assumption that the navy will use that coal, even if better fuel can be got. The item of cost is not so important as that of quality; for this affects two vital elements of naval efficiency—the speed of vessels and the distance which they can traverse without re-coaling. It would be the height of folly to make our navy take its fuel from a single colliery. In time of war, the least superiority of one fuel over another might decide the fate of a fleet; and I venture to say that our government would take the best fuel it could find, whatever became of its colliery-investment in Alaska. This supreme requirement of quality at any price distinguishes the navy from ordinary customers.

But, quality being assured, cost becomes important; and a rival fuel, of high caloric efficiency, namely, the oil of California, is now competing with coal on the Pacific coast. The relative cost of the two fuels is worth considering.

Alfred H. Brooks, the distinguished member of the United States Geological Survey, to whom we owe so much of our accurate knowledge of the resources of Alaska, estimates the actual cost of mining in the Bering region at $2 per ton. To one who has had long experience in the Pennsylvania anthracite-region, this figure seems very small. It must represent merely the actual cost in labor and supplies of mining a ton of coal from a colliery completely developed and equipped, without

any allowance for interest on the capital invested, or for accidental delays and expenses, or for special exploring-work, or for "strikes"—or, above all, for faults and folds in the coal-bed itself, necessitating new and expensive dead-work, beyond the amount required to prepare a given level for exploitation. Such underground irregularities have ruined many a promising coal-mining operation; and they are characteristic of the Bering field. Mr. Brooks seems to make allowance for them by intimating that the Matanuska coal may prove cheaper to mine, because it "appears to be less folded and faulted." But the Matanuska field is some 200 miles from port; so that the certain disadvantage of the railway-haul counterbalances the possible advantage of the present apparent greater regularity of the beds. In fact, Mr. Brooks puts the cost of Matanuska coal $1 per ton higher than that of Bering coal. But even according to his very low figures for the latter (based on a product of 1,000,000 tons per annum, railroad-freight of only $0.66, ocean-freight of $0.85, unloading-charges of $0.15, and profit to the mine-operator of $0.15 per ton), Mr. Brooks says that Bering bituminous coal should be delivered at the wharf in Seattle at $3.91 to $4.10, and Bering anthracite at $5 per ton.

Now the paper presented at the San Francisco meeting, October, 1911, by Mark L. Requa,[2] declares that "comparing California oil with Alaska coal, it is apparent that oil has complete control of the field." Mr. Requa deems it "safe to say that California oil will dominate the fuel-market on the Pacific coast during the present century, and probably far into the next century." He shows by detailed figures that coal must sell at $3.50 per ton at Douglas Island, Alaska, in order to compete with oil at 60 cents per barrel (the present price being only 30 cents) at the well. Mr. Bain himself admits that "the railroad and the leading coast-industries have now been converted to burn oil imported from California." In other words, a large coal-mine opened at the present time in Alaska, by the government or anybody else, could not be made to pay, unless it were supported by government contracts, appropriations, or privileges—in which case, it would simply be an arbitrary and arti-

[2] Present Conditions in the California Oil-Fields, *Trans.*, xlii., 837 to 846 (1911).

ficial interference with an established industry of California. From the standpoint of "conservation," the case is still stronger. The superabundant oil-supply of California is irreparably running to waste. Why should this treasure be sacrificed for the sake of the premature development, by artificial support, of a competitive industry?

It may be replied, that the United States Navy must burn coal. But I notice that the latest battle-ship launched by the Japanese is so constructed as to burn either coal or oil at will; and I fancy that we can do whatever the Japanese can do, to meet this problem—especially when our own country is producing and wasting an immense amount of oil, the "conservation" of which would be directly in the line of our most modern "emphasis."

In short, if the government needs fuel for the navy, it had better prepare its ships to burn either coal or oil, and then let the citizens of the country furnish, in free competition, the fuel found most suitable and economical.

Mr. Bain's suggestion of an army engineer, or an engineer from the United States Bureau of Mines, as an available manager for this government colliery and railroad, may be dismissed with the simple comment that such officials cannot be spared from the work they are now doing, without putting others in their present places. Consequently, there would be nothing saved by employing them—besides which, it is not likely that Mr. Bain's recommendation would determine the appointment at Washington of the occupant of such a position.

The notion that the proposed colliery could be utilized as a means of testing new methods, or investigating important questions, suggested by the United States Bureau of Mines, is open to two serious objections:

In the first place, no colliery at which experiments of that kind are continually going on, can be kept in operation, as Mr. Bain declares this one should be, in conformity with the conditions of actual practice.

In the second place, no new questions in coal-mining should be tested in Alaska, where the conditions of the industry are, in many respects, certain to prove abnormal. There are hundreds of collieries in the middle West where such novelties can be—and, if plausible, would be—tested, with results far more

useful, because more applicable to the average conditions of the industry. One colliery could make place for one promising experiment; but any one colliery doomed to try all the experiments proposed would be nothing but a station, maintained by Congressional appropriations covering its annual deficit, and justified only by its technical, not its commercial, results. Such a station might conceivably be a good thing; but, even in that case, it ought not to be located in Alaska, where results would be less valuable and more costly than anywhere else.

Mr. Bain's further proposals of leases to corporations, and a selling-syndicate, including the United States as a member, to control prices, prevent excessive profits and promote social justice, have been carefully elaborated. They are evidently suggested by the policy of the German Empire, which has, in several instances, become a partner in "trusts," in order to be able to control their operations. Concerning these plans, I offer here but two suggestions:

In the first place, the German system, which has its manifest advantages, contains also safeguards against abuses. While I do not personally believe that, as a whole, this "paternal government" is better than our American system of individual enterprise and free contract, I acknowledge the right of others to prefer it. But of this I am sure: that, if the German system is to be imitated at all, it must be copied as a whole, and not piece-meal. We must extend over the whole national service the provisions for preliminary education, secure tenure of office, and pension upon retirement, which we now apply to our army and navy only. They work well, on the whole, in the army and navy; but it does not follow that they could be wisely extended to all Federal functions and offices. This is a question too wide for discussion here. But, until we have deliberately adopted the "bureaucratic" and "paternal" system, it will be folly to adopt single parts of it.

Mr. Bain's paper presents an instance in point. He has devised a careful scheme, which we will assume to be very wise. But every feature of that scheme requires action from Congress, and any amendment of it by Congress may destroy its utility entirely. What security have we, that Congress would not thus fatally "amend" his plan?

Under this head, our best guide will be the record of what

Congress has done for Alaska already. The Act of May 28, 1908, "To Encourage the Development of Coal-Deposits in the Territory of Alaska," is a specimen of the legislation to be expected with regard to a distant Territory, and an industry concerning which the members of Congress have necessarily little knowledge. This Act, I believe, was based upon the recommendation of a Commission, appointed by Executive authority; but, in its final form, after sundry amendments, it contained features not proposed by that Commission. Sec. 3, for instance, reads as follows:

"That if any of the lands or deposits purchased under the provisions of this Act shall be owned, leased, trusteed, possessed, or controlled by any device permanently, temporarily, directly, indirectly, tacitly, or in any manner whatsoever, so that they form part of, or in any way effect any combination, or are in any wise controlled by any combination in the form of an unlawful trust, or form the subject of any contract or conspiracy in restraint of trade in the mining or selling of coal, or of any holding of such lands, by any individual, partnership, association, corporation, mortgage, stock-ownership or control, in excess of 2,560 acres in the District of Alaska, the title thereto shall be forfeited to the United States by proceedings instituted by the Attorney-General of the United States in the courts for that purpose."

The exact meaning of this verbose provision is not clear; but it has been generally understood to mean that even the holding of stock in any other Alaska coal-mining corporation by a stockholder of a corporation which had purchased coal-lands under this Act, would forfeit the title to those lands. At all events, it is plain that for the least infraction of this vague prohibition, or for any act, not supposed at the time, but determined afterwards by a judge or jury, to be such an infraction, the punishment would be the ruin of the whole enterprise of which the guilty party may have been only an official, agent, or stockholder. We know that stockholders, in consequence of the acts of their directors, managers, or otherwise authorized representatives, may have to pay damages and penalties. But the notion that the doom of immediate, complete ruin should be visited upon a corporation for *bona fide* acts of its agents, subsequently construed to be in violation of a statute, seems to the average stockholder to be confiscation, rather than regulation.

Such drastic measures, however, carry their own cure. They may catch and plunder a few unfortunate investors already

involved; but capital has one supreme, conclusive advantage in such a contest: namely, it can run away; and if, fortunately, it has not yet become inextricably involved, it can decline to go in—and there you are!

But the wisdom of Congress did not stop with the measure I have described. The Act of 1908 was really intended, by the Commission which recommended it, to furnish a way in which coal-claims sufficiently large for commercial operations could be legally acquired in Alaska. In provisions not here quoted, it authorizes individual locators under the previous law to consolidate their small claims so as to make tracts not exceeding 2,560 acres in size, and to perfect entry and acquire title to such tracts. This "encouraging" permission, however, was coupled with conditions not previously known to the law. One of these was Sec. 3, already quoted; but, as if that were not enough, Sec. 1 contained an annoying, though probably futile, stipulation as to the stockholders of any corporation thus created; and Sec. 2 gave to the United States government the right to take, for the use of the army and navy, at a price fixed by the President, the coal mined upon the granted tract—the owners being authorized to sue in the United States Court of Claims for an additional price, if they thought they could get it, and were willing to risk the expense of trying.

Of course, such conditions were not attractive to capitalists. At all events, they could not be expected to invest money in Alaska coal-mining, under such risks and restrictions, if there were another way in which such an enterprise could be inaugurated, free from these newly-devised impediments. Such a way was open under the Federal coal-land law, extended by Congress, June 6, 1900, to cover Alaska, which permitted the location of a claim of 160 acres by an individual. A locator under that law, after complying with the prescribed conditions (which need not be recited here), and obtaining his patent, could do what he liked with his own property. He could open a mine on it, or sell it to his neighbor, or contribute it for the formation, together with the similar contributions of his neighbors, of a tract large enough for a long-continued business, such as would invite the investment of capital. By simply waiting until individual patentees were ready to combine in this way, a corporation could acquire their rights for a consolidated

enterprise, free from all the burdens and restrictions of the Act of 1908.

The only way (if there was any way) to induce investors to accept the unfavorable conditions imposed by the Act of 1908, was to shut out the alternative offered by the Federal law, extended over Alaska in 1900. It is funny, but true, that this little alternative was overlooked entirely by Congress. The Act of 1908 provided simply an optional procedure, and repealed no previous legislation. Consequently, the former law is still in force; and those who do not like the conditions of the new law can still make valid locations, if they find opportunity, under the old one; so that all former real or supposed evils are continued. In fact, many locators under the former old law are quietly holding on to their rights; and new locations under that law are now and then still recorded. With Mr. Bain's declaration that these locations, whether old or new, should be promptly adjudicated and, if free from fraud, carried to the issue of patents, I heartily agree. But the fact remains, that Congress, with the best intentions, botched the whole business; and the presumption is that Congress would do so again if it got a chance. Congress should not be too severely blamed for legislating in a hurry about matters of which it knew nothing. More blameworthy, indeed, has been its failure for years to remove the incubus imposed by government upon Alaska. But my present point is, that Mr. Bain ought to feel sick at heart, in contemplating the probable fate of his thoughtful plan at the hands of Congress, and the probable shape in which it would emerge from such handling.

Of course, Mr. Bain is not responsible for that result. He has a clear right to suggest any ideal scheme which seems to him desirable. But we others have a similar right to criticise that scheme as impracticable, without even discussing its theoretical merits. I confess, however, that those who make this criticism ought to suggest, if they can, some practicable alternative; and therefore I conclude these desultory remarks with such a suggestion, not based upon adequate personal knowledge, but deserving attention as an alternative to Mr. Bain's costly and otherwise questionable plan.

From my study of sundry reports and maps, I infer that Katalla is the best port for the shipment of Alaskan coal; and

I learn that this port and its terminal facilities are now occupied by the Alaska-Pacific Railway & Terminal Co., which has already spent about $500,000 in its enterprise. This company has no connection with the wicked Morgan-Guggenheim syndicate, owns no present or prospective coal-locations, and is simply holding on to its present franchises and constructions until the Bering coal-field shall be allowed, in some way or other, to produce coal. When that desired event shall arrive, this company will be able to transport Bering coal to Katalla by a haul of from 17 to 25 miles, and therefore, at a much lower rate per ton than any other projected line can offer. At present, it is useless to prosecute either this or any other railroad enterprise, while the United States forbids the production of coal. But if coal-mining were permitted in the Bering field, this line, already surveyed and located, and provided with terminal facilities, could be extended immediately to that field. After that, the contest between Alaska coal and California oil could be settled, as all such issues have to be settled, by business competition. No doubt there will be a growing local market for the Alaska coal; and, in the fight for a market, private coal-operators will take greater risks than any manager of a government enterprise would dare to take. Meanwhile, the government would buy its navy-fuel wherever it saw fit; and Congress would be relieved of unaccustomed and incomprehensible functions.

I should add, that I do not know the exact condition, or the personal promoters, of the Alaska-Pacific Co., above referred to. Any other railway enterprise in Alaska, free from the suspicion of "conspiracy," would serve my purpose of illustration equally well. But this one seems to fill that purpose most completely, having the shortest line to the Bering coal, and having been, moreover, rarely mentioned in the voluminous proceedings of the Congressional Committee which harrowed and sifted the affairs of Alaska. Such scanty notice from such a tribunal is almost a certificate of merit!

WILLIAM H. HAMPTON, Katalla, Alaska (communication to the Secretary *):—On the basis of long observation and intimate personal knowledge of Alaskan conditions, I desire to say

* Received June 12, 1912.

that the great necessity of Alaska is facilities for transportation, which, to a considerable extent, must precede the development of the region in other respects. Capitalists are not eager to open and equip large mines, unless they can be sure of the railways which will bring their machinery and supplies, and carry away their products. On the other hand, pioneer railway-lines are not attractive investments, unless they can be reasonably assured of a profitable business from the new territory which they penetrate and develop. There is thus a dead-lock, which is, however, more apparent than real, if only let alone; for, as experience has shown everywhere on the Pacific slope, the modern American system of running railways into undeveloped regions (as distinguished from the old system of connecting by rail the centers of trade already established) is sure to be set in operation, if the opportunity be given to it. But in Alaska, the United States government has made this apparent dead-lock real, by placing artificial restrictions upon both railways and mining. And, the development of Alaska having been thus made at present impossible, various experimental remedies, such as government leases, or direct government operations, are suggested to relieve the situation which the government itself has created. In other words, new experiments are proposed to cure the evils caused by old ones.

Now, I do not think it is wise to try such new experiments in a new country like Alaska, which needs all possible fostering encouragement in the exploration and exploitation of its resources, and cannot bear any additional risk or burden. Nor can I see that such extraordinary measures would be necessary, if certain obviously just and proper things were done.

1. Let the laws be obeyed by the officers of the government, as well as by citizens; and let the citizen who has obeyed the laws be free from arbitrary interference, and, above all, from the stigma of fraud, placed wholesale upon an entire population, as a pretext for the denial of rights conferred by statute. Congress has invited the citizens of the United States to engage in the difficult enterprise of settling and developing Alaska upon certain conditions. Those conditions have been fulfilled by many citizens in good faith; and the United States has received $360,000 in payment for coal-lands, located under the law in the Bering field. Yet not a single United States

patent has been issued in that field. Some of the claims are disputed by the government, it is true, and on the merits of that dispute I say nothing. But with regard to all of them, disputed or undisputed, patents are arbitrarily withheld; and claimants are informed that the money they have paid in good faith cannot be returned to them except by special Act of Congress. When, if ever, that special Act shall have been passed, they may get their money back, without interest—for Uncle Sam pays no interest on "claims." Apart from all questions of law, is this the way to encourage the settlement and improvement of a new—and especially an Arctic—territory?

2. But these claimants may have some legal rights—a matter yet to be finally determined. Meanwhile, no new titles can be granted in the Bering field until these have been finally either confirmed or extinguished. Hence, the obvious duty and policy of the government is to settle these titles as soon as possible; and, if this duty were promptly discharged, there might be no need of other and dubious means to break the dead-lock.

3. Legislation for the prevention or regulation of "trusts" and "monopolies" is all right. Only, let it be legislation, and not arbitrary Executive interference. Let the law itself fix the conditions to which individuals and corporations are to be subject, and let them not be left dependent upon Executive orders and Department regulations, fixing new conditions. I know that it is already the privilege of a citizen to resist illegal or extra-legal regulations, and to insist upon his rights under the law. But the privilege of fighting the government is too expensive for a poor man—and rich men are notoriously unwilling to buy, together with a right, the risk and odium of a fight. Moreover, the poor man is forbidden even to sell his chances of battle before he has won a victory!

4. In the Matanuska coal-field, very little money has been actually paid to the government for coal-lands. Most of the coal-claims, I believe, have been canceled. This explains the special attention lately called to the subject of government leases for the development of that field, and in connection therewith, the plan of "unloading" upon the government, as a trunk-line, the old Alaska Central railway. A bill has been already introduced in Congress, providing for the assumption

of this enterprise by the United States, and the transfer, from the Isthmus of Panama to Alaska, of the machinery, material, and men required to complete the railroad! If it be really wise or necessary to undertake such an enterprise, the government ought certainly to be free to choose the place for it, and not be forced to select this field by having excluded, through its own unnecessary and still revocable action, the Bering field.

5. Besides the known high-grade coal of the Bering and Matanuska fields, all good, bad, or indifferent coal-lands, with an area of 12,000 sq. miles, have been arbitrarily withdrawn from exploration and location. Within this area whoever mines a ton of coal will be pursued as a criminal by United States "special agents." A still larger tract, established as a "forest-reserve," is an additional bar to the industrial progress of Alaska and the enterprise of its citizens. No wonder the situation thus created, and prolonged for years through Executive action and Congressional inaction, calls for relief. But the proposition of relief through further interference does not commend itself as wise.

6. Evidently, Mr. Bain has not been on the ground in person; I therefore beg to correct an error into which he has fallen, concerning distances, routes, and connections. Namely, the line of the Copper River & Northwestern railway, for the transportation of coal from the Bering field to its terminus at Cordova, passes at Katalla the terminus of the Alaska-Pacific Co.; and its extra length, beyond Katalla, is not 58 miles, as stated by Mr. Bain, but 75 miles in an air-line. The latter road strikes the first Bering coal 17 miles from its terminus and harbor; and at 25 miles it is in the heart of the Bering field. From a point on this direct, short route to the coal, a cross-line of about 26 miles would connect with the Copper River & Northwestern for purposes of interior trade. Having personally made, before the advent of the C. R. & NW. Co., the survey of much of the line now occupied by that company, I make this statement upon positive knowledge.

7. In conclusion, I beg to repeat that, in my judgment, what Alaska needs, and has a right to expect, is simply fair play, such as heretofore has been given to other Territories. Mr. Bain suggests that his plan of a government mine, a government railway, government leases of coal-lands, and a selling-syndi-

cate, of which the United States shall be a member, be applied to Alaska only at present, and afterwards, if the results prove satisfactory, extended to the remaining coal-lands in the States. As a citizen of Alaska, I would urge, on the contrary, that this complicated experiment be tried first on some patient not already moribund. If the results should be satisfactory, and if Alaska should be permitted meanwhile to recover strength without the interference of doctors, she might be able to stand the new treatment, when it had been elsewhere demonstrated as successful.

As to the general policy of leasing public mineral lands, I would only say that, if it should be adopted, I think the Canadian system might well be followed. Under that system, the government gives something, in the way of the efficient administration of justice, the protection of property, public improvements, etc., for the royalty it exacts. The mere acquirement of rent or royalty, without such compensations, is nothing but the old Spanish colonial system, which promotes tyranny and ends in ruin.

Flameless Combustion.

Discussion of the paper of Carleton Ellis, p. 612.

ARTHUR H. ELLIOTT, New York, N. Y.: We have all been very much interested in the results of Dr. Bone's work, which Mr. Ellis has presented to us this evening. The high temperatures attained by this method perhaps afford us an explanation of the temperatures reached in the puddling-furnace with its crude and almost primitive methods of burning fuel, by means of which we have, nevertheless, been able to melt iron in a comparatively short time, and keep it in a molten condition down to a high degree of purity. No doubt many of us have wondered, as I have done, how such a result could be obtained in such a way; and it occurs to me that, perhaps, Dr. Bone has supplied the explanation in disclosing the effect on combustion of burning gas and air after passing through, and in contact with, a porous refractory body. May not the fettling of the puddling-furnace have, in some degree, an effect upon the combustion like the influence of the porous diaphragm employed by Dr. Bone?

C. A. MEISSNER, New York, N. Y.: I have been much interested to observe the shortness of the tubes of the boiler and of the hot-water heater shown by Mr. Ellis, especially in connection with his statement that these tubes showed no deposit of scale around the point of combustion, and no deterioration. In view of the quick and complete combustion of fuel under the Bone system, and therefore the strong concentration of heat at one point, these facts, if borne out, present some very interesting features.

ANTON EILERS, New York, N. Y.: I should like to ask Mr. Ellis if any experiments have been made in the direction of applying flameless combustion in large metallurgical operations, especially as to its possible use in the very large reverberatories used in copper-smelting. If not, I would suggest that experiments in this direction might be expected to be very important.

WILLIAM KENT, Montclair, N. J.: Mr. Ellis's paper contains many interesting facts; but I do not think it is necessary to have a new theory of "surface combustion," increased chemical capacity, or catalytic action of the surfaces to explain the phenomena that he describes. Whenever a combustible gas, such as methane, is intimately mixed with air in the right proportions, so that each particle of gas is in intimate contact with enough oxygen for its complete combustion, and the mixture is heated at any point to the igniting-temperature, it will burn with the rapidity and completeness of an explosion. No kind of combustion can be more perfect, and it cannot be improved by contact with the surface of any solid material. The porous diaphragm shown by Mr. Ellis merely acts like the fine wire mesh of the Davy safety-lamps; that is, it prevents the retreating of the flame back into or behind the diaphragm so long as the latter is below the igniting-temperature, but as soon as the diaphragm gets heated on the flame side to the igniting-temperature, the flame will retreat and burn in the rear of the front surface, and among the particles of the diaphragm itself.

In regard to the rapid rate of transmission of heat through boiler-tubes when they are filled with sand, among the particles of which the gas is burning, the rapidity of transmission is

not due to any new kind of combustion, but merely to the heat-conducting power of the sand in direct contact with the metal of the tube. The sand constitutes an "extended surface," and it acts precisely like the projections of the inner surface of the Serve tube which was used to some extent in English boilers several years ago, or like the spiral metal "retarders" in contact with the inner surface of the boiler-tube, which are occasionally used in fire-tube boilers.

It is not conceivable by any one familiar with steam-boiler practice that the Skinningrove form of boiler can ever come into common use. While the rate of transmission of heat through the tube-surface may be from two to four times that in ordinary boilers, the cost of the boiler will be more than four times as great as that of a water-tube boiler of the same extent of heating-surface, or a water-tube boiler of four times the heating-surface can be built for less money. Besides this, the cost of power to push the gases of combustion through the sand-filled tubes would be an insuperable objection to the commercial use of such a boiler.

Charles Edward Lucke, New York, N. Y.: It was not my intention in coming here to speak of my own work in this field, but rather to learn of any new developments that might be reported by the lecturer of the evening. However, it might be well to review some of my old experiments and apparatus in the burning of explosive gaseous mixtures that seem to disclose all that has been described as the work of Professor Bone, and to offer a more simple and adequate explanation of the phenomena. Some years ago (about 1900), the possibility of operating internal-combustion engines by a different plan from that used by Otto and Diesel was subjected by me to a somewhat lengthy study. The scheme that seemed to offer many advantages consisted in compressing air, burning fuel in the compressed air at constant pressure, and subsequently expanding the heated products of combustion, a plan which, of course, would include the gas-turbine as one form. It became clear very early that, to get the maximum work from the volume of air compressed, this air should be made to combine with the maximum amount of fuel, or that the combustion should take place with air and fuel in combining proportions. With fuels

in the gaseous or vapor form, such combustion would be explosive; and so it seemed necessary to investigate the problem of burning explosive gaseous mixtures continuously localized in a so-called combustion-chamber or furnace. Review of U. S. patents revealed many burner- or furnace-constructions intended for the purpose, and practically all of them were tried, as well as a good many others designed by me; but all failed to work properly. These failures led to a study of the literature of explosive gaseous combustion, and practically everything in print was examined, with a view to discovering some basic principles for the operation of the burner- or furnace-apparatus. With the help of this literature research, combined with the experimental experience so far gained, I developed a method of burning explosive mixtures, derived from both gas- and oil-fuel, in localized zones under perfect control, and the means I employed were similar to those described by Professor Bone and repeated by Mr. Ellis. It is most interesting, however, to note that the reasoning that led to the result, and the explanation of the phenomena, are different from what they offer; and, of course, it is natural that I should think my own the more satisfactory.

Great stress has been laid by Professor Bone on the catalytic agency of a porous incandescent surface itself, as a means of promoting combustion, and as an explanatory theory of the action taking place in the apparatus used by us. There can be no doubt of the accelerating tendency of hot bodies on the rate of combustion-reactions. Every one who has worked with fires, especially gas- and oil-fires, knows that more fuel can be burned in a furnace after it is heated up than when cold. But, as an explanation of the peculiarities of the explosive combustion processes, this is entirely inadequate. The true explanation, as I have previously pointed out, must be based on adjustment of velocities, velocity of flow to velocity of propagation of inflammation. When porous beds or plates of solid matter are used to affect or control this velocity adjustment, and become hot in so doing, of course, they then bring into play any catalytic or otherwise accelerating tendencies that belong to them; but the prime action still remains one of adjustment of flow-velocity to rate of propagation. In free flow-paths it is always true that the flame-cap or surface of combustion will localize or remain fixed only where flow-velocity is equal to rate of propagation,

or at the place where the mixture approaches as fast as it can burn back towards the source of supply.

Let us consider the line of reasoning that leads up to this conclusion and furnishes the explanation of all these examples

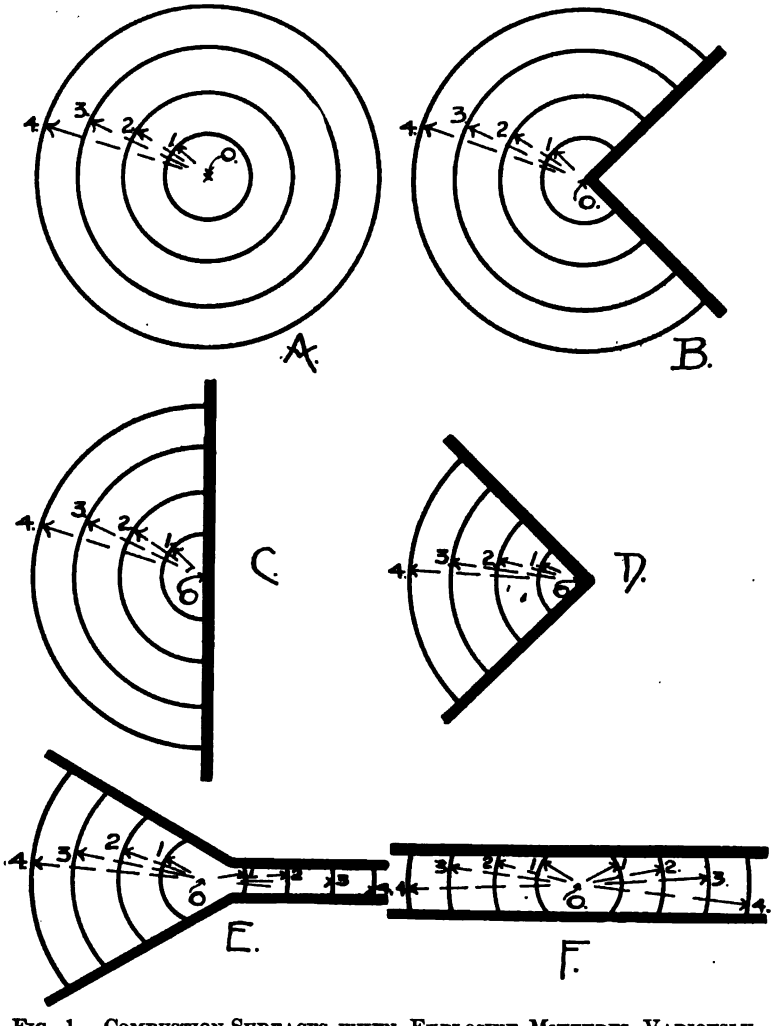

FIG. 1.—COMBUSTION-SURFACES WHEN EXPLOSIVE MIXTURES, VARIOUSLY CONFINED AND AT REST, ARE IGNITED.

of explosive combustion, starting, first, with the burning of masses of explosive mixture at rest, and later with streams of mixture flowing through various forms of passages. Imagine Fig. 1 a mass of mixture at rest, ignited at the point *O*, and

variously restricted by limiting walls; at *A*, no limit whatever; *B*, ignition at the apex of a solid corner; *C*, ignition on a flat plate; *D*, inside the apex of a cone; *E*, ignition at a point inside of a cone connected with a tube; and *F*, at the middle of a tube alone. In all cases the inflammation started at *O* will move at equal velocities in all directions and at a rate generally termed the rate of propagation and designated as *r* ft. per sec. At the end of 1 sec., combustion will be taking place on the surface of a sphere whose radius is *r* ft.; and this spherical surface, whether complete or interrupted by any boundary-walls, will separate burnt from unburnt gases. Each successive second the spherical surface of combustion will increase in radius by *r* ft. until all the mixture has been consumed. These successive spherical surfaces will be equally distant when the rate of propagation (*r*) is constant; but should it increase, the distances between the spheres of each successive second will increase, and inversely for a decrease of *r*. There are many known causes of variation in the rate of propagation that need not be taken up here, except to note that the maximum rate is that of the detonating wave studied by Berthelot, Mallard, and Le Chatelier, and more recently by Dixon, while the minimum rate is always zero, and the normal rate for any mixture may be reduced to zero by sufficient cooling, or accelerated by heating.

However the rate of propagation may vary, the combustion will remain on a spherical surface if the mixture be homogeneous; but non-homogeneous mixtures, or mixtures in irregular chambers exerting variable influences on the rate of propagation, will burn on surfaces that differ from spheres only as these influences determine. The influences that control the velocity of propagation are, in the class of cases under discussion, due mainly to heating and cooling of the mixture; and these are illustrated in Fig. 2, showing the mixture at rest in tubes of various kinds. In the first tube, *A*, the walls are supposed to be absolutely non-conducting and of the same temperature as the mixture, so that the successive combustion-surfaces corresponding to each second of elapsed time after ignition at *O* are truly spherical and equidistant. Tube *B* is provided with a hot jacket on one end and a cold jacket on the other, affecting the temperature of the mixture next the walls. On the cold

side, the edges of the spherical surface of combustion are bent back, due to cooling, while on the hot side they are bent forward, without affecting the distances between successive surfaces at the center. If the hot side be warmed to the temperature of ignition, of course the flame-travel will be wholly changed, as each point of the wall becomes then a new igniting-point, acting the same as *O*. Should the whole mass of mixture in the hot end be uniformly hot, and that at the cold

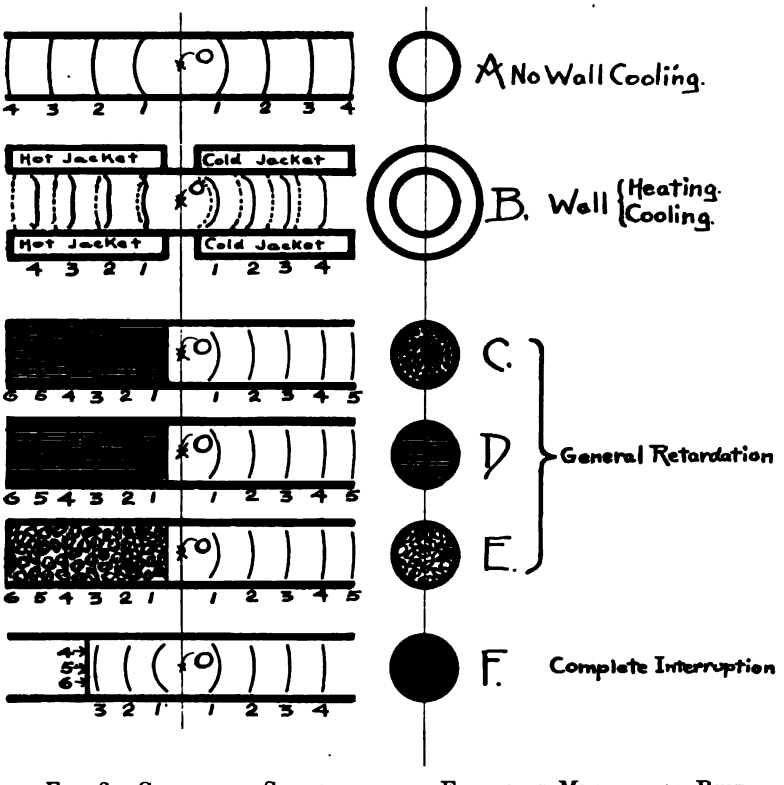

FIG. 2.—COMBUSTION-SURFACES WHEN EXPLOSIVE MIXTURE AT REST IS IGNITED.

end uniformly cold, then the successive surfaces would be unequally spaced, as indicated by the dotted lines, without bending at the angles. Tubes *C*, *D*, and *E* are non-homogeneous, having in the left-hand end packing of various kinds; *C*, a bundle of small tubes or rods; *D*, flat plates close together; and *E*, granular solid material. These all exert cooling influences on the flame passing between, and retard the whole

advance, as is indicated by the lesser distances between the surfaces on the packed side, compared with the free side. In all these cases, if the gas-passages be small and the quantity of solid matter separating them be large, then the cooling influence may be so great as to prevent flame-passage at all, by reducing the rate of propagation to zero; and one common application of this principle is the wire gauze partition, illustrated in Tube *F*.

Should the solid matter forming the multiple flow-paths for the passage of the flame be heated, it will exert an accelerating influence on the rate of propagation, and not only cease to be a flame-interrupting partition, but actually promote flame-passage.

With this preliminary discussion on the movement of the surfaces of combustion in quiet mixtures, at rates determined by heating and cooling influences in the flame-path, and the effect on the position and shape of the surfaces, let us consider an explosive mixture in motion, having a flow-velocity of v ft. per sec. as it issues from a single orifice, and subsequently changing velocity, as illustrated in Fig. 3. In the first case, *A*, the mixture flows through a parallel-wall tube at a velocity, v, equal to the rate of propagation, r, so that, after ignition at the outlet, the flame-surface remains suspended in space, the mixture being supplied just as fast as it can burn. At *B*, the velocity of flow, v, is less than the rate of propagation, so that the combustion moves against the flow-current, or back-flashes, at a rate equal to the difference $r - v$, while at *C* the velocity of flow is greater than the rate of propagation and the surface of combustion moves with the current or blows off at a rate equal to the difference $v - r$. In blowing off, however, the jet of mixture, as it moves through the atmosphere, diffuses at the edges, so that the cross-section that remains explosive grows smaller, while all gas that is torn off by diffusion is lost for combustion entirely.

That the flame-cap or surface of combustion may be kept steady, or localized, requires very accurate adjustment of flow-velocity to rate of propagation, which, to be practical and useful, must be automatic. I, therefore, early adopted the broad principle of supplying mixture at a velocity greater than the rate of propagation, with subsequent reduction of flow-velocity

by sidewise expansion. This was first tried as in *D*, where a conical nozzle was relied upon for sidewise spreading of the jet and consequent reduction of velocity, but without complete success, because, except when the angle of the cone was very small,

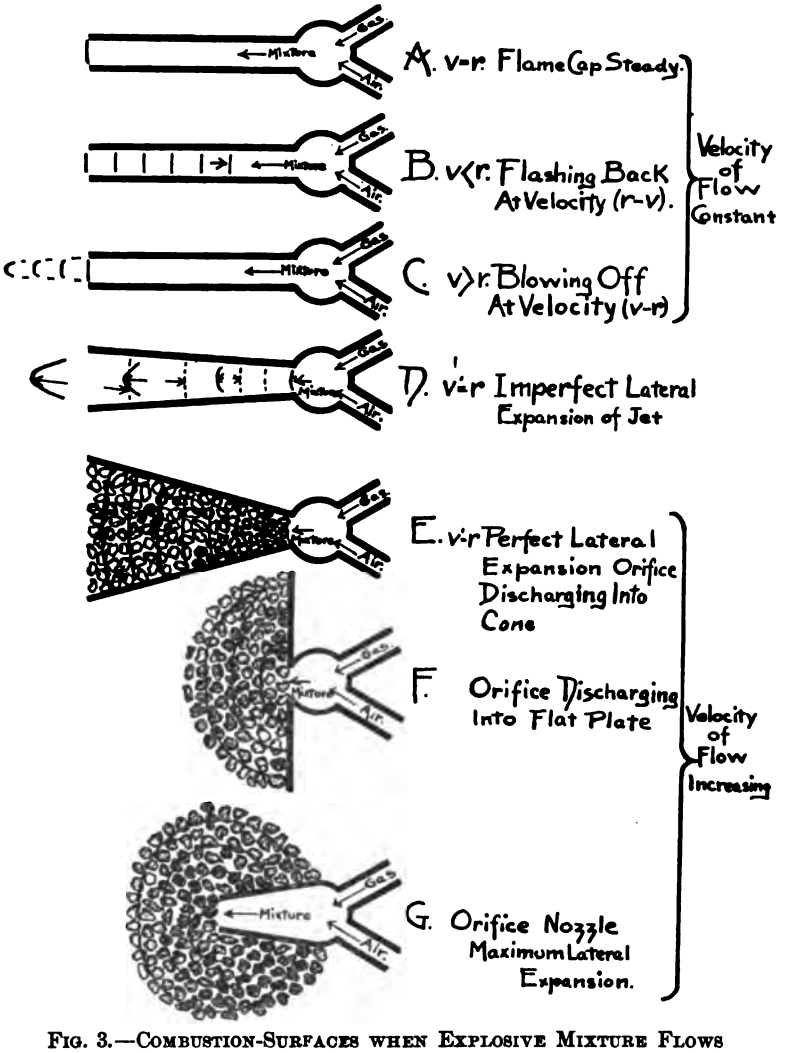

FIG. 3.—COMBUSTION-SURFACES WHEN EXPLOSIVE MIXTURE FLOWS THROUGH VARIOUSLY FORMED PASSAGEWAYS.

the jet expanded sidewise too slowly. It was expected, for example, that where the area of the cone was twice the area of the throat, the velocity of flow would be half that of the throat, so that if the throat-velocity were twice the rate of propagation equality

would automatically occur at the point where the nozzle-area was twice that of the throat. Likewise, as the throat-velocity rose to three and four times the rate of propagation, the flame-cap would locate by automatic equalization between velocity and rate of propagation where the area of cone was three and four times the throat-area respectively. What actually happened is illustrated in *D*, somewhat exaggerated. The successive flame-caps due to increase of throat-velocity located beyond the expected position, and, becoming pointed, proved insufficient sidewise expansion. This was entirely cured, even for wide angles of cone, by filling the cone with loose refractory material, as in *E*. Of course, this cut down the area available for flow, leaving only so much as corresponded to the voids, which depend on the size of the grains, being less for small and more for large grains. This worked perfectly and called into play the secondary influence of acceleration of rate of combustion as soon as the material became hot, as it did almost immediately by the combustion proceeding between the particles.

By the automatic adjustment of flow-velocity and rate of propagation the combustion-surface instantly localizes between the granules and heats them rapidly to incandescence, and then, as the rate of combustion is increased by this high temperature, the equalization-zone moves back a little, just as it would with a flow-velocity reduction; but it soon reaches a steady state. Increase of flow-velocity will cause the combustion-zone to move away enough to conform to equalization, as before; and a great range of feed-rates is possible, if the angle of the wedge be large enough, without driving the combustion beyond imposed limits. An angle of 180° is illustrated in *F* and 360° in *G*. In all these cases, it matters not at all what refractory material is used to promote the lateral spreading of the mixture-stream, reducing flow-velocity and, by heating, accelerating the rate of propagation, to bring about equalization of velocities. Nor does it matter just how the material is formed, whether as loose or bonded granules, or holes in plates, or a series of baffles, or porous baked plastic matter. In my early work broken fire-brick was always used because of its availability; and, shortly after igniting, the granules always stuck together, forming a porous lump or plate or diaphragm, depending on the conditions of forming; but the combustion of the mixture

was exactly the same in this bonded mass as in the loose granules, except, of course, when, as occasionally happened, the whole mass really fused and flowed—an action which led to the subsequent use of more highly refractory material.

In the last form shown, Fig. 3, *G*, where the nozzle projects into the jet-spreader, the zone of combustion returns on the nozzle and will heat it above the temperature of ignition, causing the mixture to ignite inside, unless the walls be thick and good insulators. To prevent any such igniting of the mixture in feed-nozzles, it is generally best to use flows of the type *F*, which, of course, may be multiplied without end to enable any desired area of fire-bed to be developed. A variety of forms of such multiple feeds are shown in Fig. 4, all of which work precisely alike in principle. The first construction, *A*, has several feed-orifices, each supplied by a branch from a header-pipe, the velocity at the exit of each into the bed being greater than the rate of propagation, and reducing, as it flows through the bed, by more or less spherical spreading. When the diameters of these combustion-spheres become greater than the distance between orifices, they join and produce a continuous wavy surface of combustion above, completely protecting the feed-orifices. Exactly the same result is attained by the holes in the partition shown in *B*, all fed from a common feed-chamber, and likewise in *C*, where the number of nozzles is multiplied considerably, reducing the size of the waves in the surface of combustion where the separate hemispheres join. In this latter case, *C*, the combustion-surface will locate closer to the orifice-plate and tend to heat it more than in case *B*—half the distance between centers of orifices approximately fixing the maximum height of combustion-surface. This makes it necessary, in using closely-spaced orifices, to form them in a refractory plate of sufficient thickness or otherwise construct the feed system to prevent the supply side getting warmed to the ignition-temperature by heat conduction.

One convenient way of doing this for some service conditions is to form the feed-orifices by using a layer of fine granulated material under the coarse, as in *D* and *E*. This layer of fine material, if loose, is supported on a grating or perforated plate, through which the resistance is less than through the fine bed, so that the velocity will be high enough on exit from the fine bed;

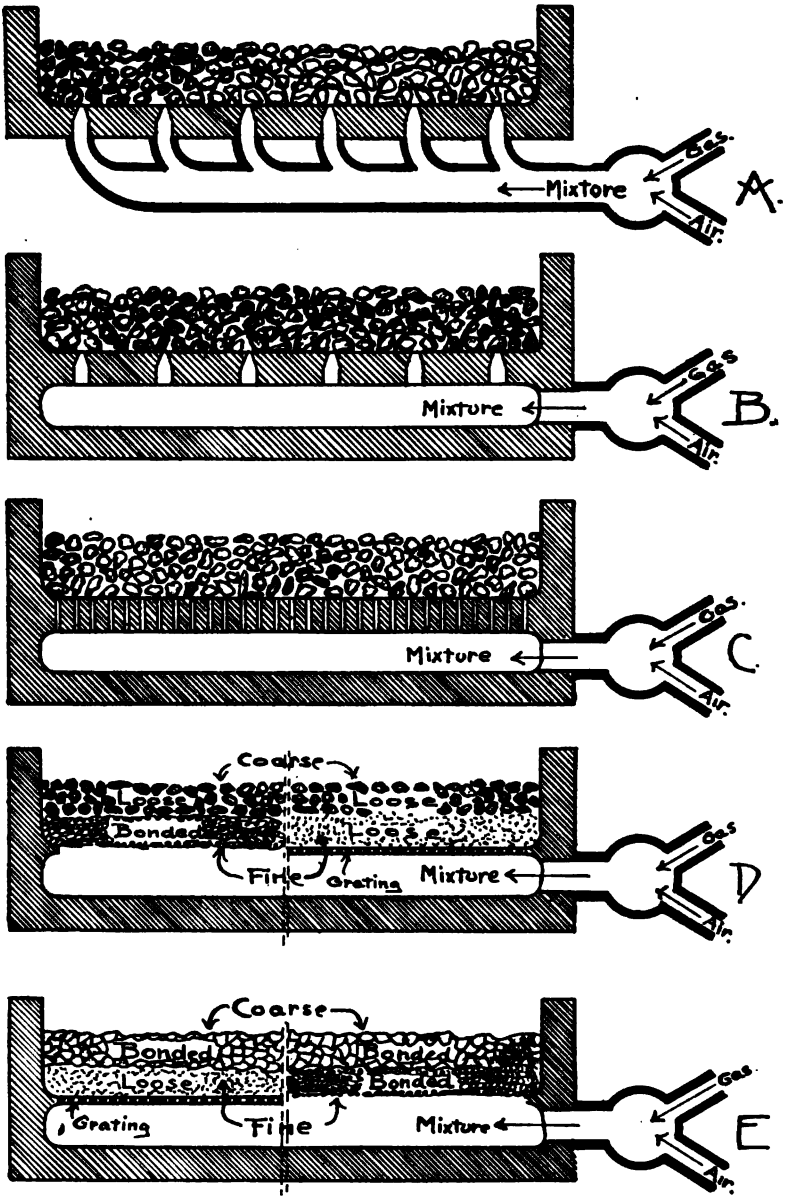

FIG. 4.—VARIOUS FORMS OF POROUS BEDS FOR THE COMBUSTION OF EXPLOSIVE MIXTURES FLOWING THROUGH.

but if bonded or formed as a porous brick, the layer is self-supporting. The high exit-velocity from the fine or small-orifice bed prevents back-flash, and the reduction of velocity in passing into the upper area prevents blow-off. Four combinations of fine and coarse beds are shown: loose above bonded; loose above loose; bonded above loose; and bonded above bonded; it being understood, of course, that by bonded is meant any porous baked plastic matter, cemented granules, or just holes drilled or formed in plates, or a succession of baffles. When the lower or fine bed or its equivalent series of feed-holes is kept cool enough, and the holes small enough, it may act like an interrupting partition to resist back-flash, but it is impracticable to depend on this; the only really reliable dependence is sufficiently high flow-velocity through the orifices. It is interesting to note that the explosive combustion may be carried out for a small range by completely removing the top layer of *D* and *E*, which is exactly equivalent to removing the bed from the feed-orifices of *A*, *B*, and *C*, if the exit-velocity be just equal to rate of propagation or only slightly greater. In this case the action becomes the same as in Fig. 3, *A*. For more severe service conditions more specific means must be used for avoiding excessive back conduction of heat towards the supply, but these are not shown here as they add nothing to the discussion of the combustion principles forming the topic of the paper.

It is believed that these principles, all of which were announced some years ago by me in papers before the American Society of Mechanical Engineers, are a clearer explanation of the working of Bone's apparatus than the speaker has offered, and considerably older.

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[NOTE.—In this Index the names of authors of papers are printed in small capitals, and the titles of papers in italics. References to papers expressly treating of the subject named are likewise in italics; and casual notices, giving but little information, are usually indicated by bracketed page-numbers. The titles of papers presented, but not printed in this volume, are followed by bracketed page-numbers only.]

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